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DEPARTMENT OF COMMERCE

UNITED STATES BUREAU OF MINES
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DEEP MINING METHODS, CONGLOMERATE MINE OF THE CALUMET AND HECLA CONSOLIDATED COPPER CO.

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BY

HARRY VIVIAN

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DEEP MINING METHODS, CONGLOMERATE MINE, CALUMET & HECLA CONSOLIDATED COPPER CO.[1]

By Harry Vivian[2]

INTRODUCTION

This paper deals principally with the retreating method of mining as applied in the Calumet & Hecla Consolidated Copper Co.'s deep Conglomerate mine, pointing out some of the difficulties encountered in deep mining and describing the particular methods used at Calumet to carry on a large-scale operation at great depth with a minimum hazard factor. The circular is one of a series dealing with mining methods and costs which is being prepared for and published by the United States Bureau of Mines.

ACKNOWLEDGMENTS

Acknowledgment is due to those officials and operating heads of the Calumet & Hecla Co. whose years of familiarity with the conditions developing at the Conglomerate mine as great depth was attained have enabled them to contribute valuable information to the subject of deep-mining problems.

BRIEF HISTORY OF THE DISTRICT AND EARLY MINE DEVELOPMENT

The copper district of Michigan is in the extreme northern part of the northern peninsula and extends through Keweenaw, Houghton, and Ontonagon counties, the northern portion projecting into Lake Superior at Keweenaw Point. The copper-bearing lodes are all found within a narrow belt from 2 to 4 miles wide and over 100 miles long, the central portion of which consists of a plateau, running in a northeasterly direction at an elevation of 400 to 600 feet above Lake Superior, from which the ground slopes down on either side, gradually toward the west and more abruptly eastward. In a number of places the plateau is cut across by valleys, the most prominent among which is the Portage Lake Gap, which offers a navigable channel for lake steamers into the heart of the copper district. The waters of Lake Superior on three sides of the region, greatly temper the climate.

Michigan was admitted into the Union as a State in 1837. The first information relative to the occurrence of native copper on Lake Superior was furnished in a report by Dr. Douglass Houghton, State geologist, submitted in 1841 to the State legislature.

In 1843, by a succession of treaties with the United States, the Chippewa Indians relinquished their claims to the lands of the upper peninsula of Michigan, and immediately upon the issuance by the War Department of Government permits to individuals to explore for

1 - The Bureau of Mines will welcome reprinting of this paper, provided the following footnote acknowledgment is used: "Reprinted from U. S. Bureau of Mines Information Circular 6526."

2 - One of the consulting engineers, U. S. Bureau of Mines, and chief engineer, Calumet & Hecla Consolidated Copper Co.

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for minerals and locate tracts of land, a wild rush commenced to the district, even before the completion of the linear survey of the public domain, which was then being made in the State of Michigan by the United States Government.

This system of permits was found to be very unsatisfactory and was abandoned in two years. Thereafter land was sold as surveyed and deeded to the purchaser by the Lake Superior Land District Office, established in 1847.

Evidences of copper mining go back to times probably predating that of the Indians, to a race of inteligent people who mined with some system on Keweenaw Point and also on Isle Royale. Masses of copper, some of enormous weight, have been found in old pits which they dug, and in some cases stone or timber platforms were found built in the pits, evidently to assist in raising the masses. Crude implements of stone and copper were usually found in great quantities. In 1844, the first actual mining operations were begun by white men in Keweenaw County, and from that date to the present time, operations have been carried on continuously.

Ninety per cent of the copper of the district has come from six orebodies, one in the Calumet and Hecla conglomerate and the other five in amygdaloids - namely, the Baltic, Isle Royale, Kearsarge, Osceola, and Pewabic amygdaloids. The Isle Royale was opened in 1852, the Quincy and Pewabic lodes in 1856, the Calumet and Hecla conglomerate in 1864, the Kearsarge and Osceola amygdaloids in 1874, and the Baltic lode in 1897. All the ore mineral is native copper, although in the upper workings some native silver was found associated with the copper.

The Calumet conglomerate orebody occurs in a felsitic conglomerate, which is traceable for a great many miles but is found in most places outside of the Calumet & Hecla property as a thin sandstone which in many places is but a few inches thick. At Calumet the bed is from 10 to 12 feet thick near the surface, widening out in the central portion of the property to about 20 feet at the 81st level, which is 8,100 feet below the surface, along the dip. Figure 1 is an isometric sketch of the conglomerate lode. Production from this lode began at Calumet in 1866, and the richest portion was found in the upper areas of the mine. The copper content of the rock remained well above 90 pounds per ton until 1885, 20 years after starting operations. Since then it has gradually decreased to about 40 pounds per ton at present.

From the beginning of operations, the amygdaloid and conglomerate types of lodes have been explored and mined by a great many different mining companies. One hundred and ten companies were actually brought to a producing stage, at a huge cost, which can not be definitely ascertained. Fourteen of these companies returned to their stockholders an amount equal to or greater than the investment, and over 50 per cent of all dividends paid were gleaned from operations in the conglomerate lode of the Calumet & Hecla Co. The total production of copper from the district to the end of 1930 has been 8,403,000,000 pounds.

At one time the copper output from Michigan was over three-fourths of the entire production of the United States, reaching its maximum in 1916 with a production of 269,700,000 pounds. The production in 1930 was 166,400,000 pounds. During the past 28 years, Michigan's percentage of the production of the United States has declined from almost 26 per cent in 1899 to a little less than 9 per cent in 1928.

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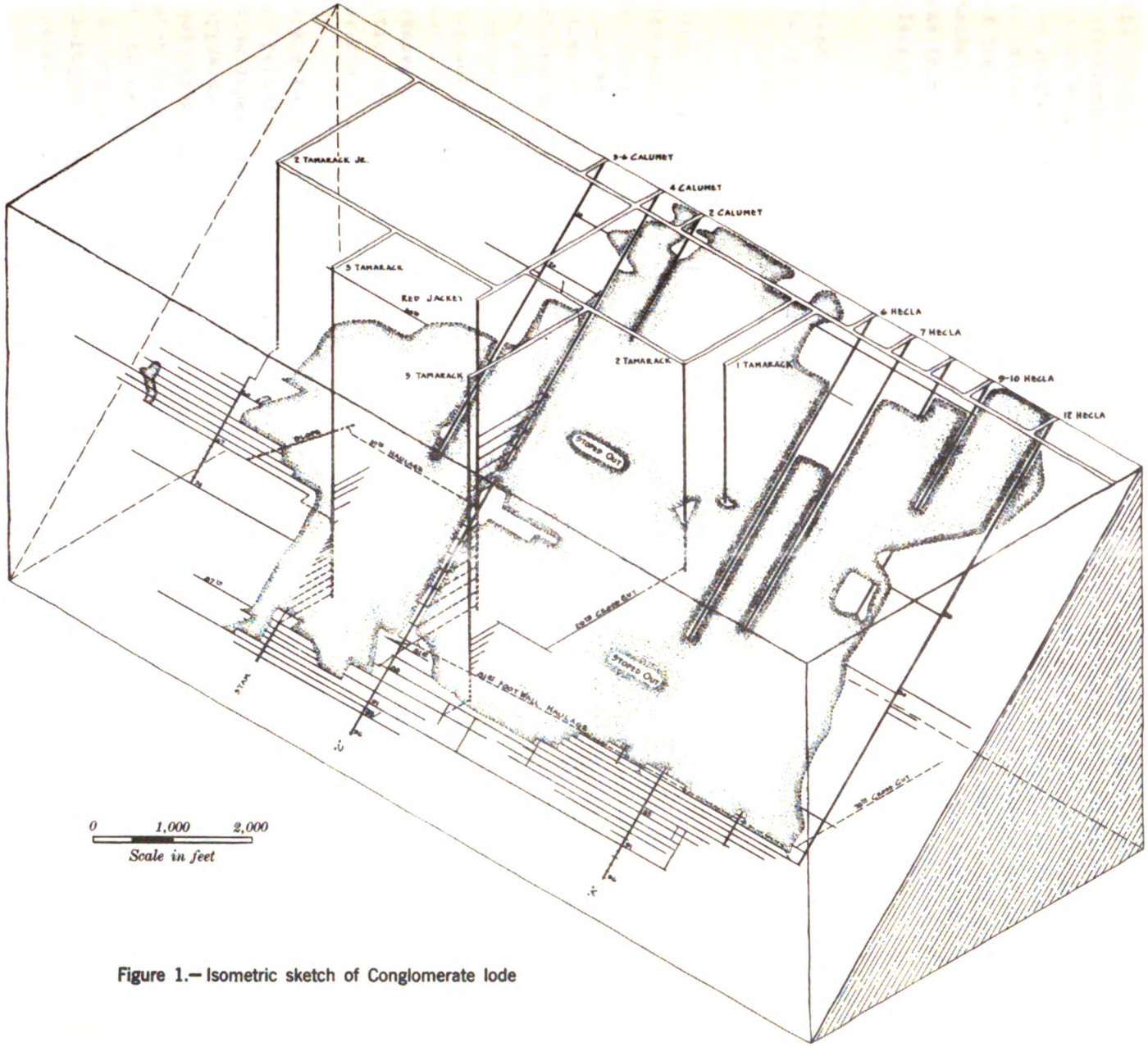

Figure 1.— Isometric sketch of Conglomerate lode

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GEOLOGY

The Keweenawan formation of Michigan consists of a series of basic lava flows with interbedded acidic conglomerates and sandstones. The beds strike with the general trend of the shore line of the Keweenaw Peninsula, which juts into Lake Superior in a northeasterly direction, and their outcrops mark the eroded edge of the southern limb of a synclinal formation which dips under Lake Superior and whose northern limb appears again on Isle Royale and on the north shore of Lake Superior in Canada. This series is separated from the Cambrian sandstone on the east by the great Keweenaw fault, which extends nearly the whole length of the peninsula and along which the older series appears to have slipped up and over on the sandstone. The fault has a slightly flatter dip toward the west than the beds of the Keweenaw formation. Throughout the series there are also numerous minor faults where the fault plane cuts across the bedding.

The entire series consists of several hundred separate flows of these basaltic rocks, having a total thickness of several thousands of feet. The copper-bearing lodes are almost entirely in the central portion of the series. Felsitic porphyries are intruded into the series in great mass at several points.

The original source of these flows is considered to have been a welling slit located somewhere near what is now the center of Lake Superior but which at the time of eruption was the crest of a gently sloping plateau. The present dip of the beds, varying from 25 to 70°, is attributed to the tilting brought about by the subsidence of the erupted mass, due to its weight, into the void formed below their expulsion point. As the lavas cooled, quantities of gas collected into bubbles and rose to the surface of the flow. Some of these bubbles were entrapped near the top as the cooling lava became viscous, forming various types of vesicular cappings, as the top crusts of the flows. These top vesicular cappings were the permeable portions of the flows, and the secondary native copper replacing the original minerals in the vesicules formed what is known as the amygdaloidal veins of the district. Most of these flows appear to have followed each other closely, but in some instances the interval between flows was long enough and other conditions were such that felsitic débris from the same original eruptive source was carried along and deposited upon the surface of some of the flows, forming the conglomerate beds of the series.

The favored theory of native copper deposition in both the amygdaloidal and conglomerate beds holds that the great intrusives underlying the series gave off solutions during the period of crystallizing which were rich in copper, arsenic, and sulphur. These solutions being expelled under enormous pressure, followed the permeable channels along the cappings of amygdaloids and through the sandy material, cementing the conglomerate pebbles together. In their upward course cooling took place, and when regions of lower pressure were reached, the solutions reacted with the highly oxidized lode rock; the arsenic and sulphur were oxidized at the expense of the hematite, which is presumed to have been formed by the oxidation of the original iron-bearing minerals of the lodes. The native copper was precipitated, replacing the original minerals which filled the amygdules of amygdaloids or replacing certain minerals forming part of the cementing material between the conglomerate pebbles.

PHYSICAL CHARACTERISTICS OF ORE AND INCLOSING ROCKS

The Calumet conglomerate is a reddish colored bed, varying in thickness from 12 to 20 feet, interstratified with the volcanic rocks of the Keweenawan series and overlain and

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underlain by thick beds of trap. The lode consists of pebbles of felsite and quartz porphyry cemented together by small particles of rock, quartz, calcite, and native copper, the rock particles in the cement being similar in composition to the pebbles. The rock is highly abrasive and tough, and has a general hardness of 7. Copper occurs chiefly in the filling between the pebbles or as part of the cementing material, although it is not uncommon to fine pebbles partly replaced by native copper, and frequently fine copper occurs within the pebbles. The bed has a general strike across the Calumet property of N. 33 E., with decidedly little variation. There are so few changes in strike that exploration drifts may frequently be kept in straight alignment for several hundred feet. The lode dips to the northwest at an angle of approximately 38° at the surface, flattening to about 36° at a depth of over a mile vertically.

Within the conglomerate frequently there are found layers of sandstone, usually lenticular in form but quite often of considerable extent. These streaks or beds, greenish gray in color, while commonly barren of copper themselves, usually occupy a position between the commercially mineralized parts of the lode and must be broken with the vein material. The well-marked parting along the bedding of the sandstone requires special attention in placing timber supports.

The entire hanging trap is checkered at various distances apart with slips or jointing faces, nearly all of which are tight and dry, and at angles varying from 30 to 70° with the bedding. Few shatter zones exist, but in places the slips are thickly grouped.

The footwall of the Calumet conglomerate is a highly fragmental amygdaloidal capping on the underlying trap. This noncommercial foot material is in general very friable and contains a large proportion of the softer minerals such as calcite, chlorite, and laumontite in which characteristics it is in contrast to the extremely hard conglomerate vein. When the weight of the hanging wall over any considerable area is permitted to concentrate upon an unmined pillar of vein material, the result is likely to be disastrous due to the thrust of this hard vein pillar upon the weaker foot material below, which causes it to burst upward into the drift opening. This condition was an important deciding influence in the adoption of a retreating system of mining.

The uniformity of the dip and strike, mentioned above, would suggest the absence of many important faults, slips or crossings. However, there are a large number of joints and bedding planes in and along the conglomerate which have a very important bearing upon mining operations, and an extensive system of slips and parting faces in the hanging which cause large blocks to cave for unknown distances above the stopes. A very comprehensive and reliable report, embodying the results of a long-time study of this particular subject, was made by Dr. W. R. Crane of the United States Bureau of Mines and was issued by the bureau as Bulletin 309, in 1929. Reference to this report is suggested for a full and detailed description of the effects of strains and stresses upon rocks, as only a few of the important facts, noticeable in this particular mine, will be included in this paper.

PRESSURE OF BROKEN-ROCK MASSES

The existence of an extensive system of jointing, together with faults, shatter zones, crossings, etc., leave little doubt that all regional strains which were originally inherent in the igneous rock, have long since been relieved, and there is little reason for assuming that any stress other than the weight of superincumbent broken rock masses, acting

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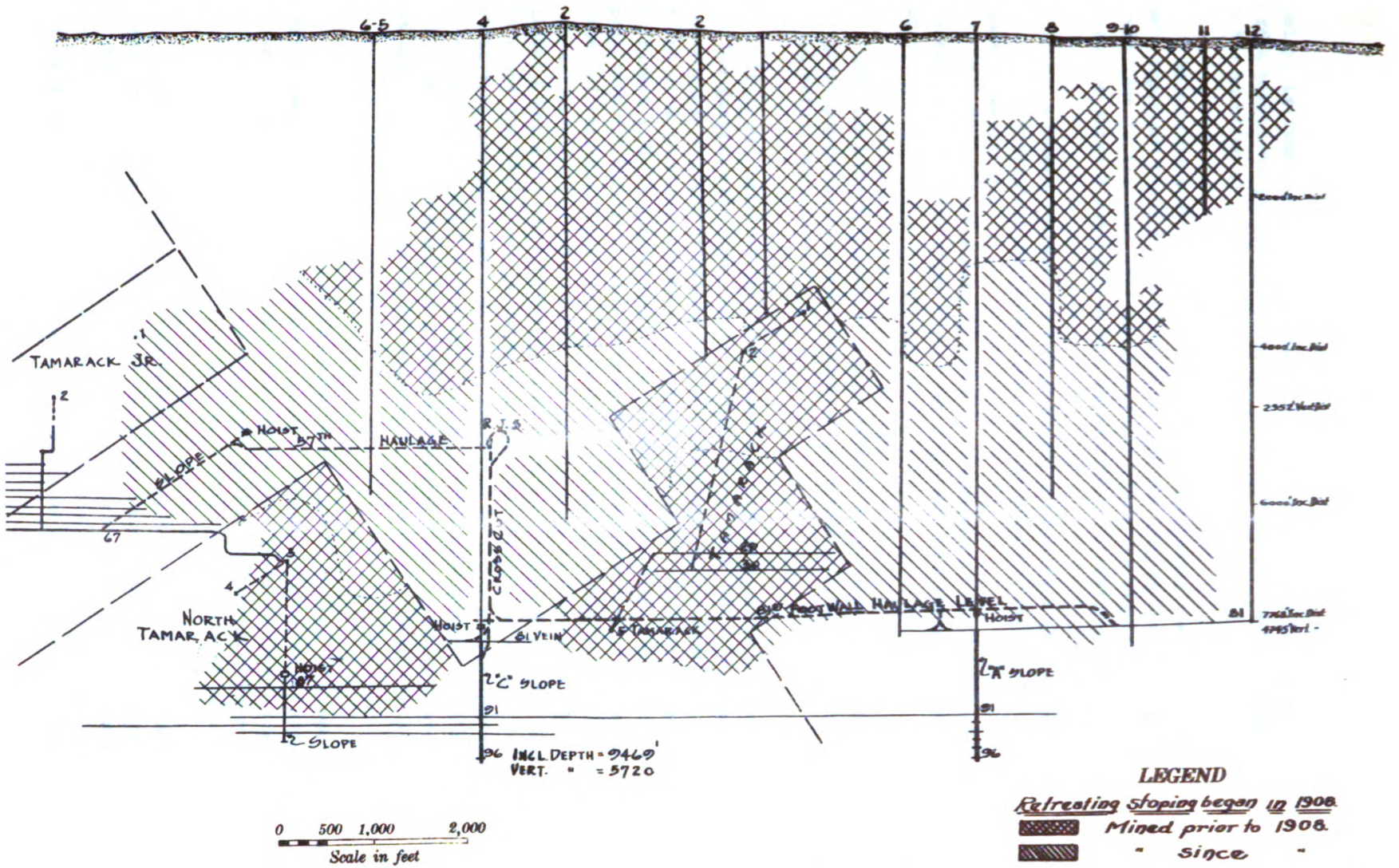

Figure 2.— Horizontal projection of the Conglomerate workings

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vertically, is responsible for the tremendous pressure which in the past has caused disastrous rock bursts and pillar failures, and finally, for depths below 3,500 to 4,000 feet has forced the adoption of a particular method of mining.

Major faults and to a lesser degree slips crossing the vein at various angles occupy a very important place in the distribution of the hanging load throughout the active working areas, due to the fact that along a fault plane the rhythmic or uniformly progressing doming action above a caving area may be seriously interferred with. This may result in:

1. A sudden increase of rock pressure upon certain pillars or temporary supports.
2. A concentration of load upon certain points along the fault walls with possible rock bursts resulting.
3. Transfer of a large portion of the support usually given by the thrust of unmined areas bordering the edges of the caved ground, resulting in concentration of a load, greater than normal, upon the active working area.

Bedding planes, joints or slips, particularly those paralleling the dip, when subject to great pressures have been observed to produce a sufficient amount of drag or creep to prevent actual rock burst.

It is not believed that any fixed rule may be applied to ground movement or pressure in the Conglomerate mine, as it is well known that new conditions are being met with daily on any particular retreating face and also that no two nearby stopes present the same conditions at a particular time. A systematic longwall retreating method has been in use in the Calumet & Hecla Conglomerate mine workings since 1908, beginning at a vertical depth of approximately 3,500 feet. This system has immeasurably simplified the problem by reducing the number of variable conditions to be met with and has made it possible to predict in advance an action about to take place. Before the acquisition of the Tamarack Mining Co. by the Calumet & Hecla Co. this property, which consists of an irregularly shaped area extending through about the center of the Calumet & Hecla property from a depth of 3,500 feet to the present bottom, was mined as a separate operation by an advancing room-and-pillar system to a vertical depth of about 5,000 feet. Figure 2 shows a projection of the Conglomerate workings. The variable conditions of pressure, movement, and rock burst were amply demonstrated by this operation, which was discontinued prior to acquisition of the property by Calumet & Hecla.

That the weight of the superincumbent mass of broken rock over the mine workings is directly responsible for air blasts or rock bursts seems to be an accepted fact locally. No idea exists as to the magnitude or even as to the limits of this force except that it is agreed that old crushed-stope areas now open to inspection fully demonstrate that nothing short of a hard, close-grained solid mass will withstand the force permanently.

Depth and the resulting increased pressure are the main causes as proved by the consistent increase in relative severity and frequency of blasts as depth is attained in all fields. While pillar failure is quite common through slabbing at depths from 2,000 to 2,500 feet vertically, actual rock-burst pressures probably do not develop above 3,500 feet on this lode. From 4,000 feet vertically special precaution must always be taken to avoid them.

The two components of the pressure from the overlying rock-mass give immediate evidence of their presence upon completion of a stope. The force normal to the dip of the

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vein causes bending and bulging of the hanging, and the force parallel to the dip is evidenced by crushing of the stull headings on the up-vein side where considerable movement takes place by an actual opening developing between the stull and its heading on the down-vein side.

It is believed that immediately above stoped-out areas the overlying rock masses become loosened along preexisting slips or parting faces. The height to which this loosening or breaking up may go depends upon the depth and upon the area mined at that depth. In retreating with approximately 100 per cent extraction of the lode, it is believed that over the extensive areas stoped in this mine, the main pressure block or dome has undoubtedly reached its maximum possible size and weight for excavation of similar thickness. It may be said that it is never possible actually to measure the height to which the overhanging rock breaks up in the immediate vicinity of a deep-stoped area. However, it has been possible in two vertical shafts at Tamarack which penetrate the vein from the hanging-wall wide, and around which the shift pillars were crushed, to gage the height above the vein by the amount of crushing and twisting which took place in the shaft timber sets. The height indicated in these two instances was between 500 and 600 feet, at a time when the shaft pillar, originally 19 feet thick vertically, had been crushed down to approximately 3 feet in vertical thickness by a squeezing out of stope pillars which might safely be said to have been practically completed.

This main pressure block is supported by the solid material at the foot of the arch or dome, both laterally and along the dip; and where a systematic retreat has been in progress over a great area, such as in the case of the Conglomerate mine, with no intervening pillars left standing, the arch may be considered to move downward as the lower support recedes, the lateral thrust following down along the shaft pillars and the upper thrust point gradually following along as the stope filling material crushes into the voids of successively lower openings.

As the pressure block moves downward, various slips and joints are affected, and upon the manner in which the lines of weakness develop depends the amount and kind of temporary support needed in the working area. Undoubtedly this dynamic pressure block at its lower extremity and along its lower sides delivers thrusts of hydrostatic nature. It is doubtful if the hanging load may ever properly be considered static over a retreating stope. The load on the temporary support is directly in proportion to the time the support has been in place for a given position. The failure of a support in one position may be the signal for the immediate failure of other timbers not necessarily adjacent thereto but at points which may be considered to be at the other end of an arched mass of disturbed rock. It may be seen from the above that various combinations of direct pressure, strains and thrusts are at work at all points, and so no exact knowledge may be had as to the nature of the material to be set in motion and no sure prediction may be made as to the distribution of the load.

The heaviest shocks heard and sometimes felt for considerable distances outside the immediate mining areas are rarely located in the mine, and it is probable that they result from the sudden breaking away of large masses at relatively great heights above the hanging wall of the vein during the doming process accompanying hanging subsidence.

METHODS OF DEVELOPMENT AND MINING

The advancing system of mining was used to a depth of 6,000 feet along the lode or about 3,500 feet vertically. While this system of mining was in vogue the practice was to

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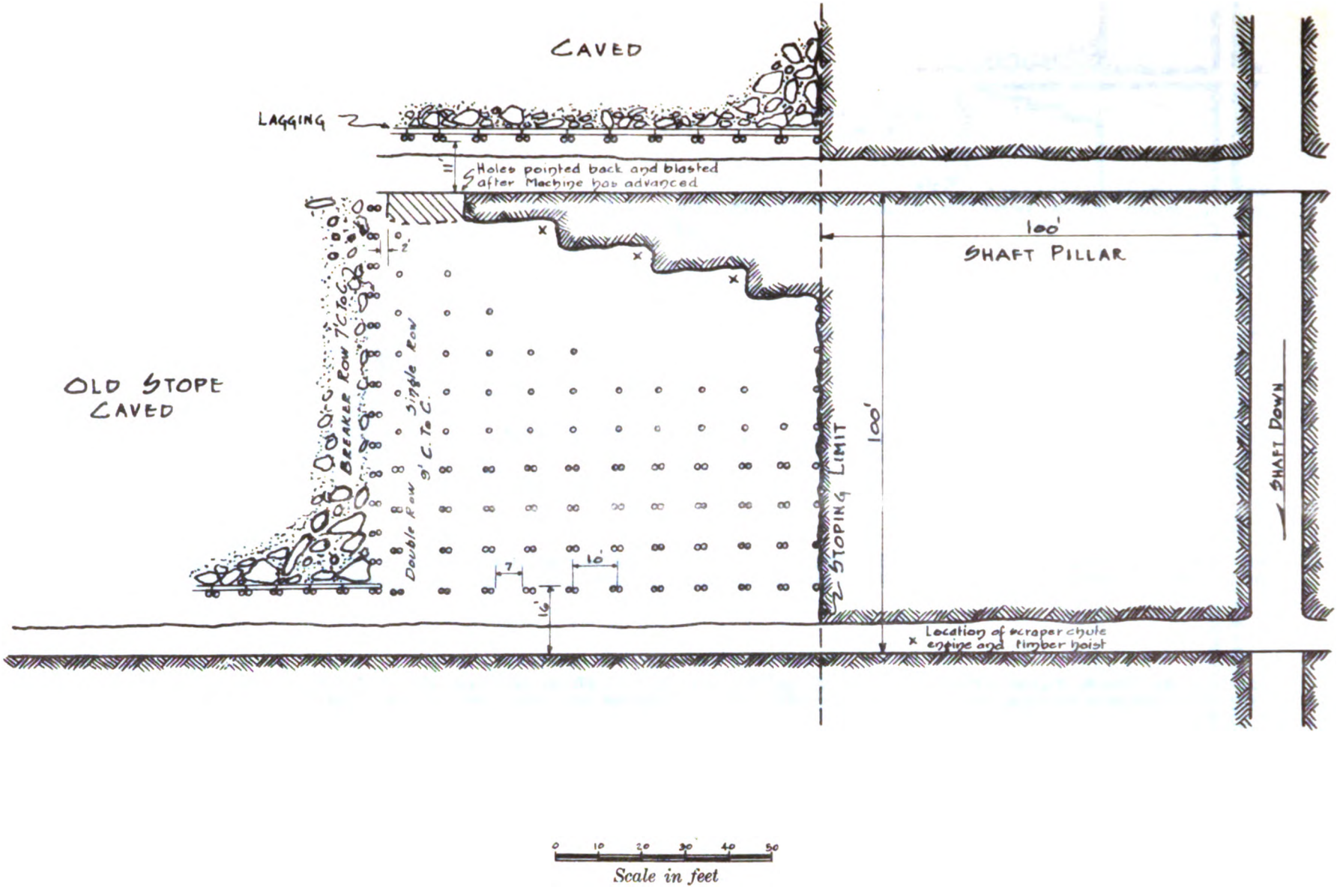

Figure 3.— Open stope, Conglomerate lode

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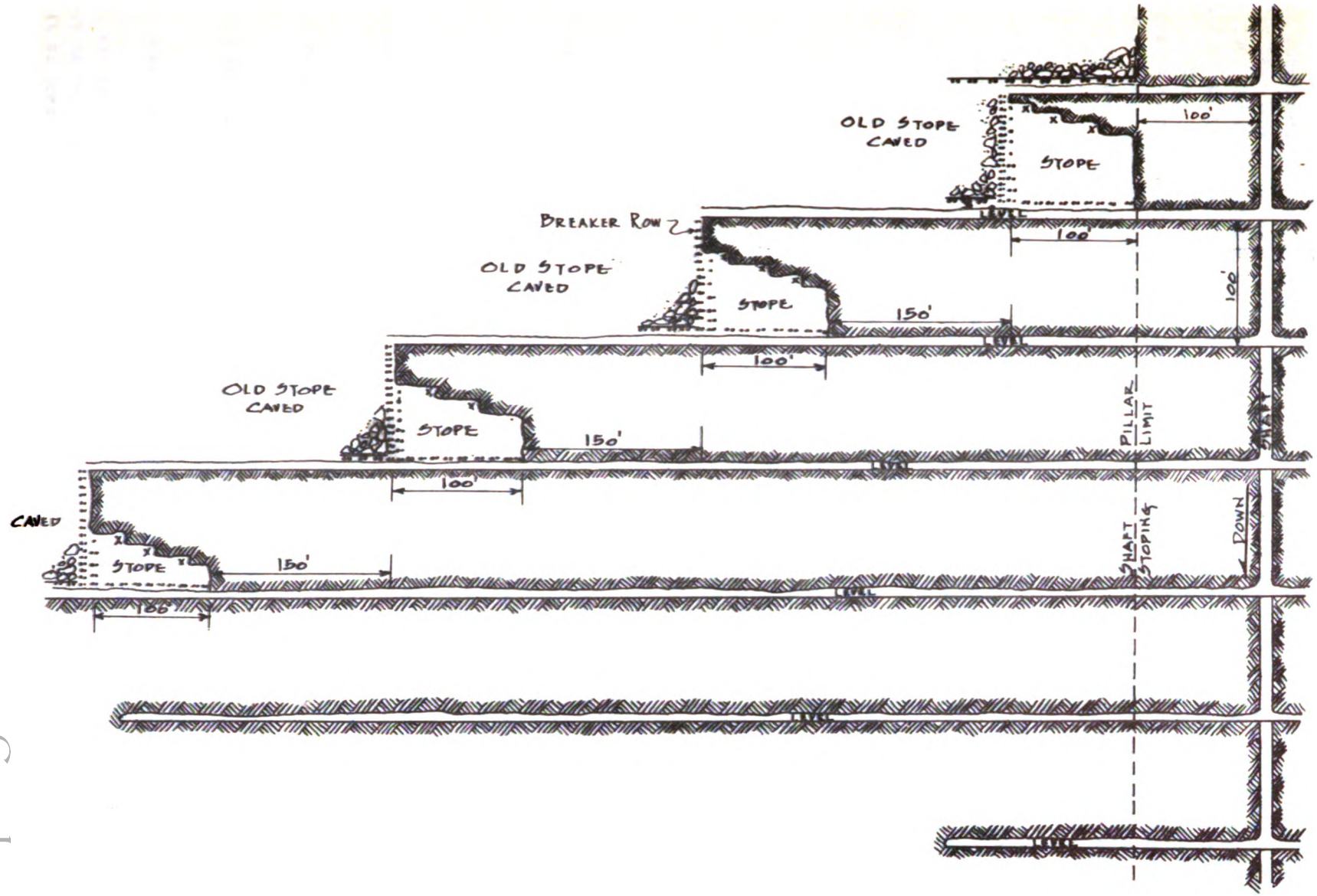

Figure 4.— Retreating stoping system, Conglomerate lode

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leave floor arches under each level and above the stope from below. These arches vary in thickness, or rather in depth, along the dip of the lode, from 8 to 15 feet on an average, and constitute lenticular rock pillars, paralleling the strike at every 100 feet in depth. They have long since been crushed by the closing in of mined-out areas above and below, and in 1909 the first attempt was made to mine them systematically. Since that time over one-half of these ancient arches have been retrieved at good profit and with practically no casualties. The closing in of the surrounding stopes has been so complete and the compacting pressure so great that the caved material exposed, as the backs are removed, frequently has the voidless appearance of virgin territory, although in some cases where a sudden breaking away of hanging took place it is necessary to timber to hold the fill while recovering floor arches.

From 6,000 feet to the present bottom stopes at 8,000 feet on the dip (4,900 feet vertically) the retreating system of stoping has been used entirely in all sections of the mine (figs. 3 and 4). This system began contemporaneously with the removal of "old backs or floor arches" in 1909, and from its inception the practice has been to remove 100 per cent of the vein material, mining the floor arches out entirely, as the retreating-stope face progresses toward the shaft pillar. It may be well to say that such a large area has already been mined by the retreating system and the recovery of "backs" has progressed so far above the present active working areas that the effect of any back support from pillars left standing under the old system of mining has long since passed entirely out of the picture. The retreating stope faces may therefore be considered to be, in a strict interpretation, deep workings operating under the maximum pressure made possible by the load of superincumbent rock. The mineralization of the Calumet conglomerate lode has been of such uniform distribution throughout the areas mined that pillars of unminable ore are almost never left standing under present practice.

In mining to greater depths, inclined subshafts have been sunk in the vein, and it is intended to leave pillars of vein material on each side of the shafts to absorb the hanging load (see fig. 1). These pillars have been about 100 feet wide, above the 81st level and are now being recovered, but it is planned to increase this width to 200 feet on each side for the deeper areas below the 81st level. Recognizing the loss due to the deferred recovery of contained mineral values in these pillars, the management gave careful consideration to the problem of establishing the main productive shafts in the footwall before deciding upon the present practice.

The practicability of such a footwall shaft depends upon several factors, important among which are the following:

1. A sufficient assurance of ore continuation to warrant the investment in barren rock work.

2. The material in which the shaft is to be located must be of such character that with a reasonable expenditure for supports it will not collapse.

3. The location of the shaft or of material suitable for the shaft must be such that connection to the vein workings through the intervening barren material will not be so long as to make the cost prohibitive, while at the same time the distance from the vein must be great enough so that during the process of removing the last stope on the vein, immediately above, the ramming effect of this last pillar shall not be sufficient to destroy the shaft or connecting crosscut.

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4. The length of time during which the shaft must be kept open.

A certain background of experience was available to call upon in the study of deep-shaft location on the Calumet conglomerate lode inasmuch as a block of ground at the north end of the property extending from the 57th to the 67th level had been mined through a shaft situated 90 feet in the footwall. This shaft, approximately 1,200 feet long, while at a considerably shallower depth than the present bottom of the mine, gave constant trouble, due to crushing of the timbers and disalignment of skip ways. Almost constant repairs have been necessary and the entire shaft has been retimbered several times. The original supports were specially designed cast-iron columns and steel I-beam wall plates carrying steel T-rail lagging over the wall plates. This form of construction proved to be very objectionable, principally due to the fact that no cushioning effect was afforded by the columns or dividing timber and that in consequence the material suddenly ruptured without any warning to afford time in which to reinforce the area. Such a condition proved very hazardous and finally the entire shaft had to be retimbered with wood.

In 1919 an experimental footwall shaft located near the "A" incline (fig. 1) was cut in an amygdaloidal lode situated 180 feet in the foot of the conglomerate. This bed was selected in spite of its relatively great distance from the conglomerate because of the homogeneous nature of the material, and its known uniformity of dip and strike, a most important element when it is realized that such a shaft must act as its own pilot during sinking operations. This shaft had been completed for about 300 feet below the 81st haulage drift and connections made to the conglomerate drifts below, when operations were discontinued because of the adverse conditions in the copper industry in 1921. Upon reopening the mine in 1922, the shaft and the immediately surrounding rock were in such a badly shattered condition that any attempt to rehabilitate it was abandoned. As no material of more suitable quality than the bed mentioned is to be found in the foot of the Calumet conglomerate lode, all plans for the future call for shafts in the vein.

The question has been asked as to the possible advantage in deep mines of sinking to great depths first, next driving development openings to the boundaries and finally applying the system of retreating the stopes toward the shafts, all provided funds were available for such extensive development. Such a plan would undoubtedly reduce the timber cost in stopes and the cost of shaft maintenance, but it is the opinion of the writer that the objections to such procedure far outweigh the advantages, particularly in thin vein deposits of material similar to the conglomerate. Some of the objections may be stated as follows:

1. It is not possible to explore a comparatively thin dipping bed for any great distance below the workings by drilling operations, due to the impossibility of keeping the drill within the vein which ordinarily carries the commercial mineralization.

2. The expenditure necessary to an extended development program together with compound interest upon the same up to the date of realization by stoping operations, is something the average mine operator as owner hesitates to face, even though able financially, due to the undeterminable value of the results to be attained and the well-known hazards of mining enterprises. To make absolute outlay against doubtful returns is the picture which usually appears, and the result is that most deep mine managers prefer to be paid as they go.

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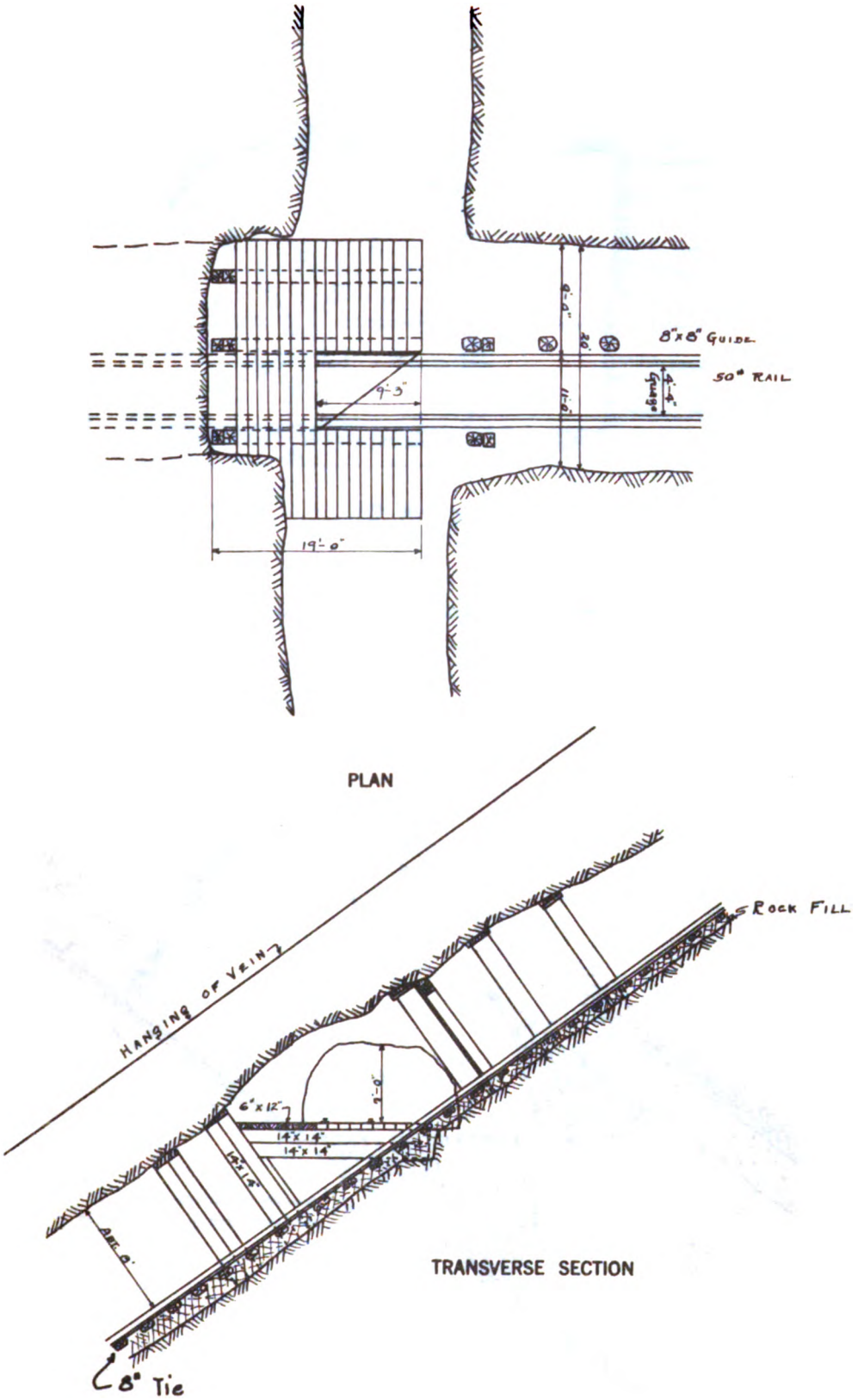

Figure 5.— Plan and section of typical plat

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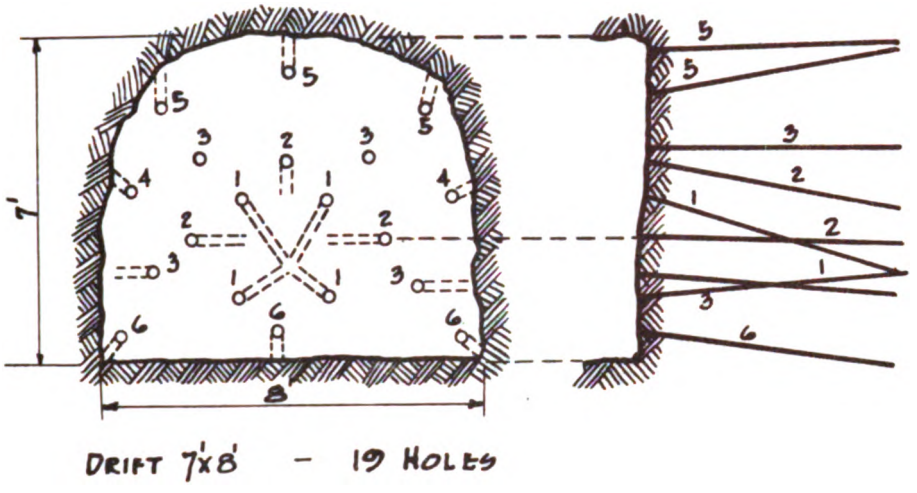

Figure 6.— Pyramid cut used on Conglomerate drifting

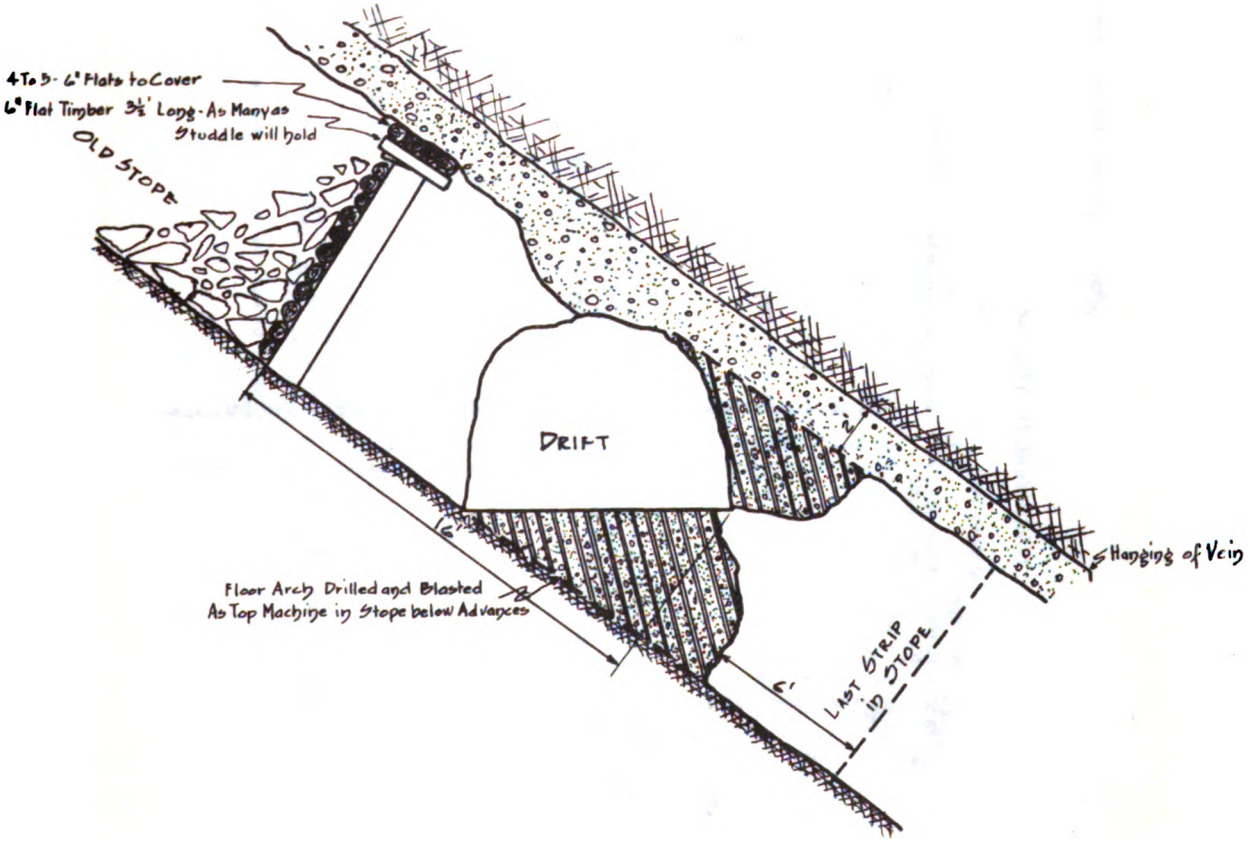
4 To 5- 6" Plats to Cover
6" Flat Timber 3½' Long - As Many as Studdle will hold
OLD STOPE
DRIFT
Floor Arch Drilled and Blasted As Top Machine in Stope below Advances
6'
Hanging of Vein
LAST STRIP IN STOPE

Figure 7.— Section showing removal of floor arch

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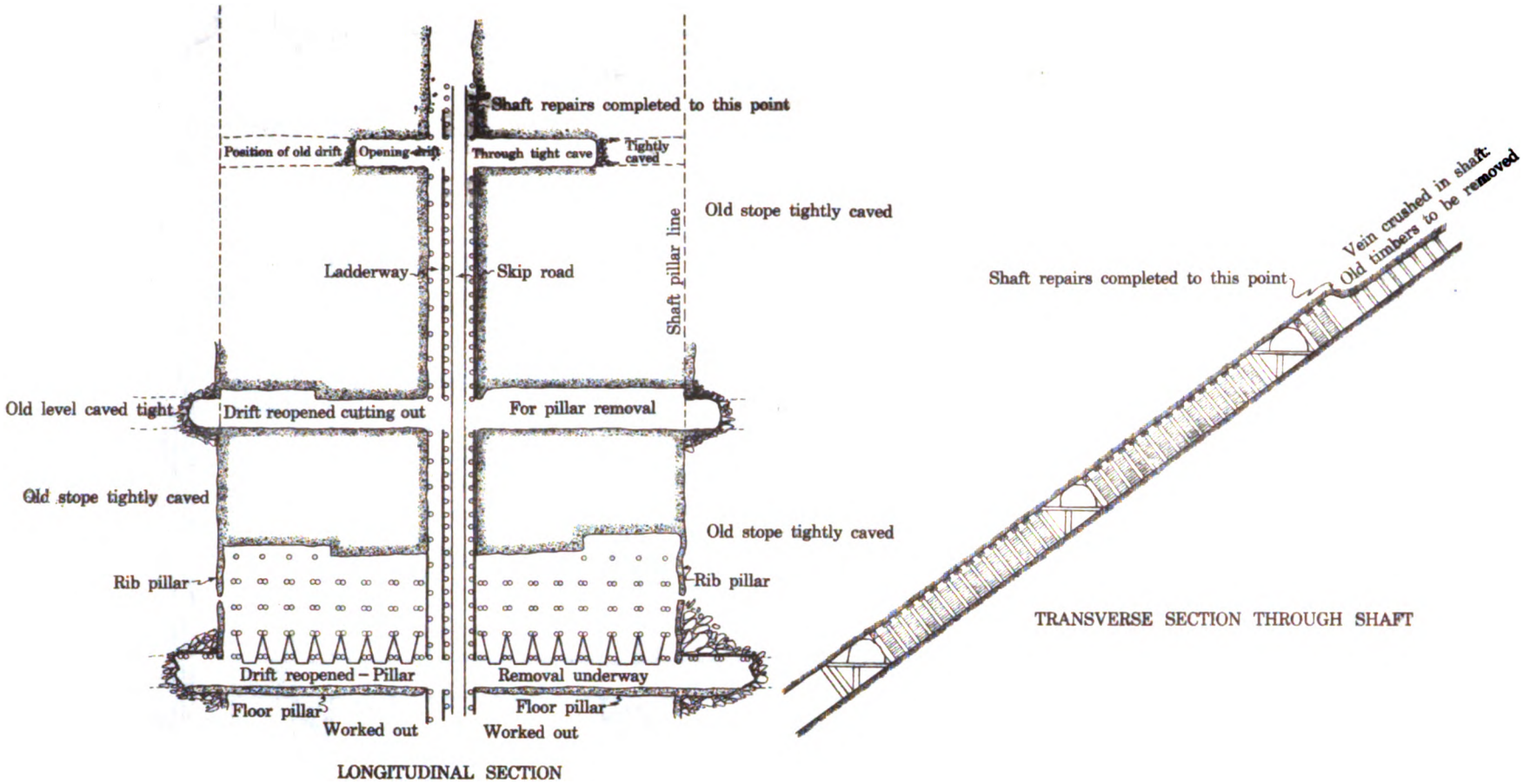

Figure 8.—Method used in removal of shaft pillars

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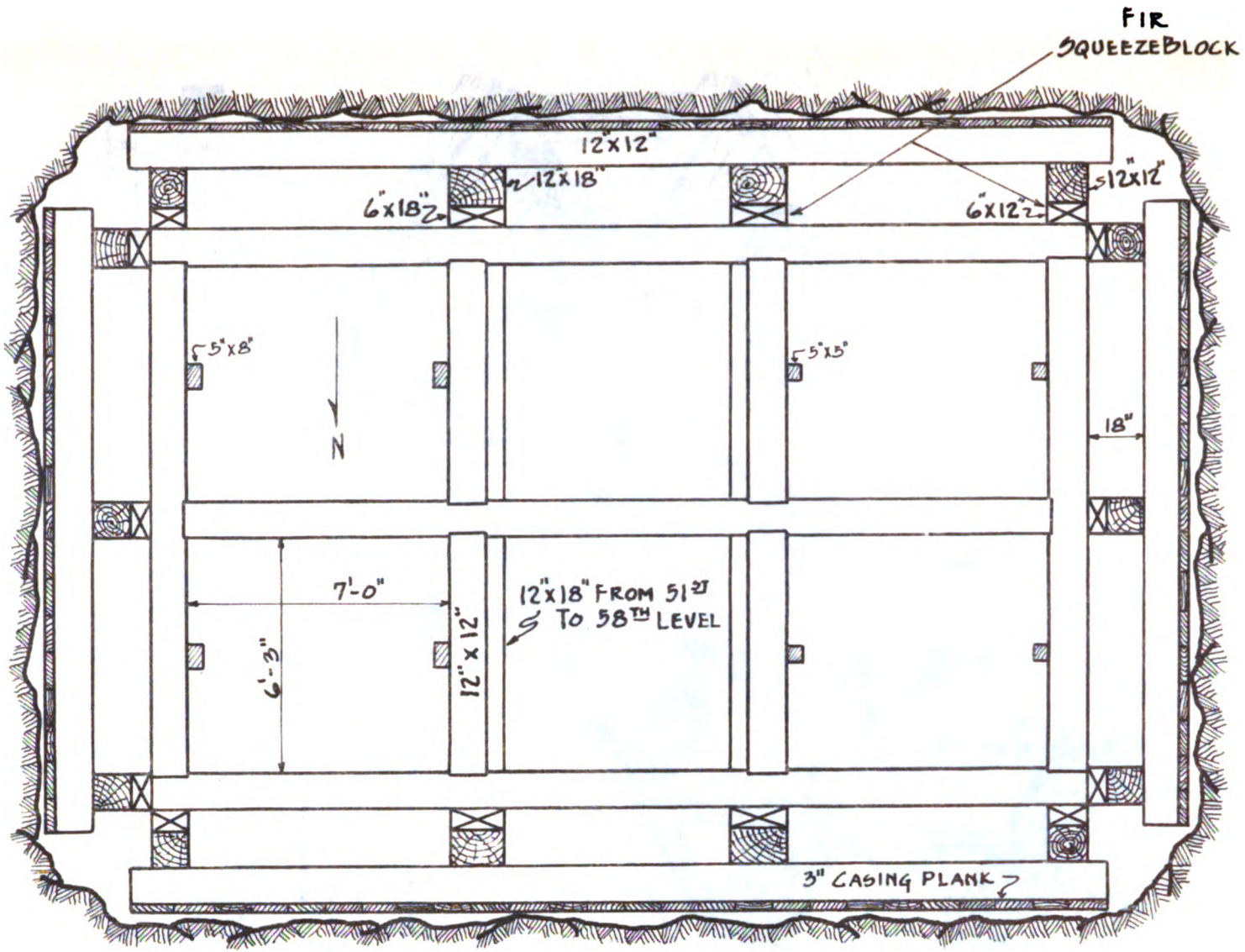

Figure 9.— Umbrella timbering in Red Jacket shaft

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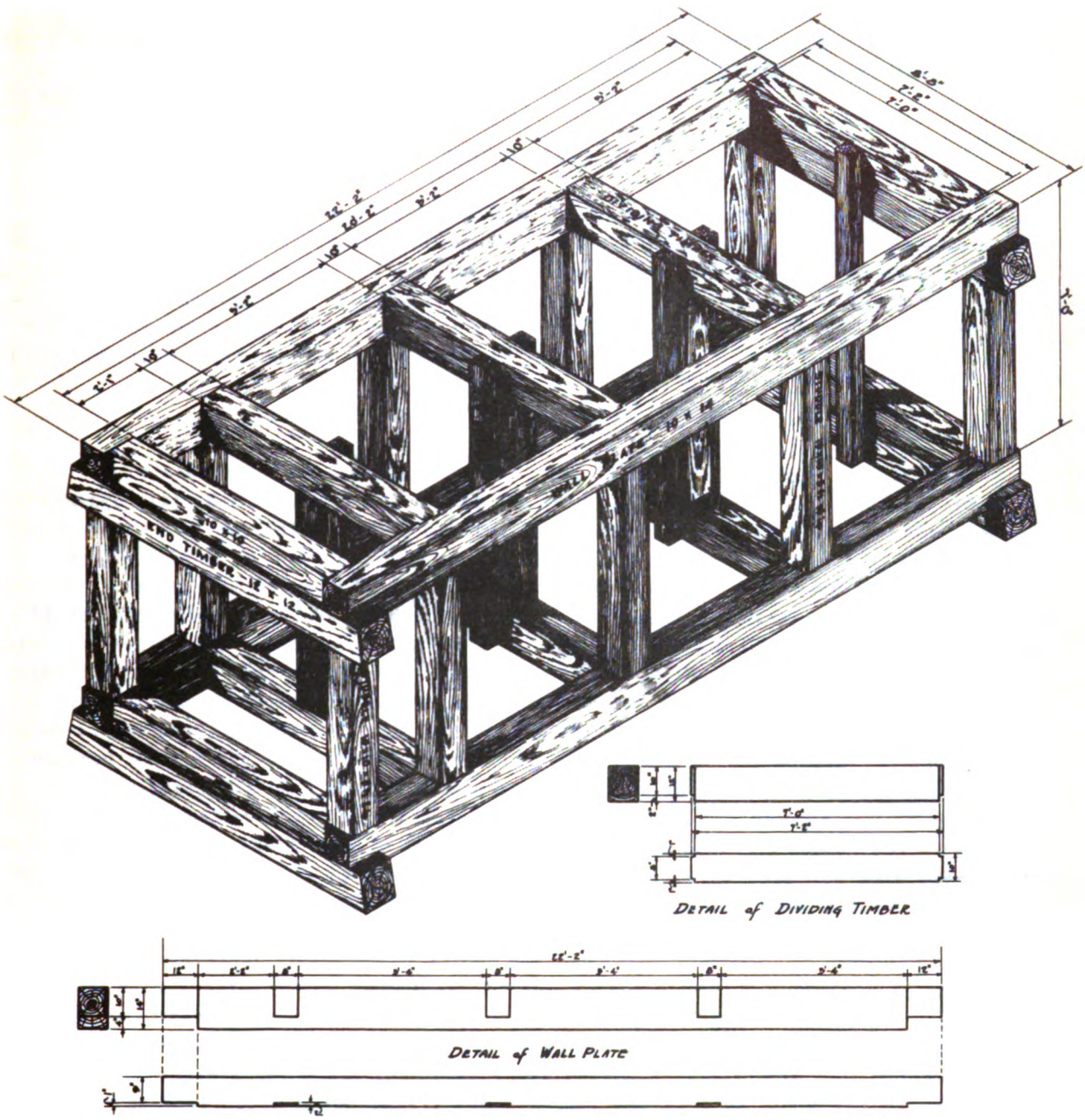

Figure 10.— Shaft timber, Tamarack No. 3 and No. 4

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3. All rock in the Conglomerate mine vein must be drilled and blasted. The overlying rock pressure may be said to offer no advantage of any kind to assist in mining the ore. That the reverse is the case is witnessed by the large number of timber supports necessary to make working places safe and also by the frequent and always possible bursts of rock from exposed faces. Serious consideration has been given to mining blocks 400 feet in depth, leaving a 400-foot block above to be mined later from the top down, after hanging pressure has been relieved by the collapse of the top stopes above, and proceeding to great depth in this manner.

SHAFT PILLARS

No rock pillars are ever left standing in the Conglomerate mine stopes. The length of the pillars of vein left standing to protect the operating shafts has been increased continually as depth was attained. In some of the early work at very shallow depths the pillars were about 15 to 20 feet long, gradually increasing to 75 feet for all active openings above the 50th level and to 100 feet from the 59th to the 81st level. It is planned to leave nothing less than a 200-foot pillar below the 81st level.

DETAILS OF SHAFTS AND 81ST LEVEL HAULAGE WAY

By reference to the isometric sketch (fig. 1) and the horizontal projection of the mine workings (fig. 2), it will be seen that the vein has been opened up by a large number of inclined shafts and several vertical shafts, which were sunk to and through the vein and connected to it by a series of crosscuts. The main production shaft is the Red Jacket, 4,900 feet deep vertically and having six compartments timbered throughout with 12 by 12 inch timber sets placed on 5-foot centers (fig. 9). The six compartments are each 6 feet 3 inches by 7 feet, set in two rows of three each, and in addition there is a narrow pump compartment. This shaft reaches the vein just above the 57th level, and at this point, as well as for about 400 feet above, the squeezing effect of the hanging wall load has been severe at times in the past, in spite of the fact that a pillar of vein material 500 feet square was left unmined around the shaft to prevent shaft disalignment. On account of this hanging thrust, an auxiliary set of timber has been employed in this area outside the regular sets. This set, known locally as the "umbrella set" (fig. 9), takes the squeeze and offers a ready opportunity for making shaft repairs without interruption to hoisting operations. Red Jacket shaft connects with the vein by crosscuts on every third level from the 36th down to the 81st. The No. 5 Tamarack, a vertical shaft and next in size, has four 5 foot 2½ inch by 7 foot compartments set in line, and in addition a 3 by 7 foot compartment for pipe. Twelve by twelve inch square-set timbering on 7-foot centers is used throughout. This shaft is used as the main pump shaft for the deep areas of the entire mine. Two large pump rooms, each containing three vertical triplex plunger pumps, operated by 350-hp. motors, are located just off the shaft, one near the bottom, 5,156 feet vertically below surface, the other at approximately the mid-depth point 2,424 feet deep. The bottom room connects with the vein on the 87th level (vertical depth 5,200 feet), the crosscut and part of the drift being used for water storage, which is protected permanently by leaving a block of unmined ore of sufficient size to resist pressure cracks. The upper pumps operate in series with the lower ones, and only nominal sump capacity is required.

No. 3 Tamarack, a vertical shaft, has three 5 foot 2½ inch by 7 foot compartments in line, in addition to one pipe compartment of 2 feet 6 inches by 7 feet (fig. 10). This shaft is about 5,100 feet deep and connects with the vein by crosscuts. It is used principally as a supply shaft for the mining areas at the north end of the property from the 67th

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level down and also contributes largely to the pumping requirements of the property, having two sets of electrically driven, horizontal duplex plunger pumps, one at 1,000 feet below surface and the other at 2,600 feet down. These pumps handle the major portion of the surface water entering the mine at this end.

No. 12 Hecla containing one skip way, about 7 by 7 feet and a manway about 4 by 7 feet, was sunk in the vein at the extreme southern end of the orebody. This shaft is used principally for raising and lowering men and as a supply and equipment handling shaft. It is bottomed on the 81st level and is now the only inclined shaft in operation from this level to surface. In all other inclined shafts, above the 81st, pillar-removal operations are in progress, retreating from the bottom toward the surface. The 81st level is considered to-day the top of a new mine in process of development, and the extensions of two slope shafts below this point are separately designated as "A" and "C" slopes (fig. 2). In line with the crosscut from No. 3 Tamarack to the vein, a third slope shaft is started below the 87th level. This slope is connected with "C" shaft and served by the 81st level haulage to Red Jacket shaft. Thus all parts of the property are connected to the 81st haulage way, and it may be well to describe this project a little further.

Eighty-First Level Haulage Way

As mining progressed to depths between 7,000 and 8,000 feet on the vein (4,900 feet vertically), the crushing load concentrated upon the inclined shaft pillars became such as to cause frequent interruption to hoisting operations. Large repair crews were required at all shafts, and maintenance work was necessary during the hoisting shift as well as between shifts. In the case of the Conglomerate mine, when ground movement began, such tremendous weight of hanging-wall material was involved that pressures reaching the "rock-burst" point were encountered. The exceedingly hard and abrasive conglomerate lies directly upon a highly brecciated lava top or amygdaloid, the amygdaloidal fragments containing considerable calcite, chlorite, and other soft minerals, while the material cementing the fragments consists of a very friable rock of fine texture, much resembling volcanic ash. The thrust on the Conglomerate mine shaft pillars is to a great extent absorbed by this less resistant foot material, which loses shape during the "squeeze" and bulges into the shaft on the footwall side, causing the worst form of shaft disalignment (fig. 11). Because adequate protection is impossible against a movement of such intensity, and because there is such a tremendous investment tied up in the millions of tons of shaft pillars, the management was prompted to study possible ways by which some plan of operations might be found which would permit realization upon these pillar reserves, would reduce shaft maintenance charges, and at the same time would assure the ability to prosecute mining to still greater depths unhampered by limited and uncertain shaft capacity. The proposition of sinking a vertical shaft in the South Hecla area similar to the Red Jacket shaft was unattractive because of the trend of mineralization toward the north and the consequent narrowing down laterally within the property lines.

As a result the 81st-level project was undertaken after acquiring the Tamarack lands, which occupy the center of the Calumet conglomerate area, as shown on the map attached (fig. 2). It may be considered, in a way, a reestablishment of surface facilities at a depth below surface of 8,100 feet down the dip. In principle, it is a scheme for concentrating at the 81st level all main haulage requirements for a mine below this level, locating on the 81st the collars of all shafts penetrating below this point, and making possible the ultimate use of existing vertical shafts for rock hoisting and other purposes, so that continual,

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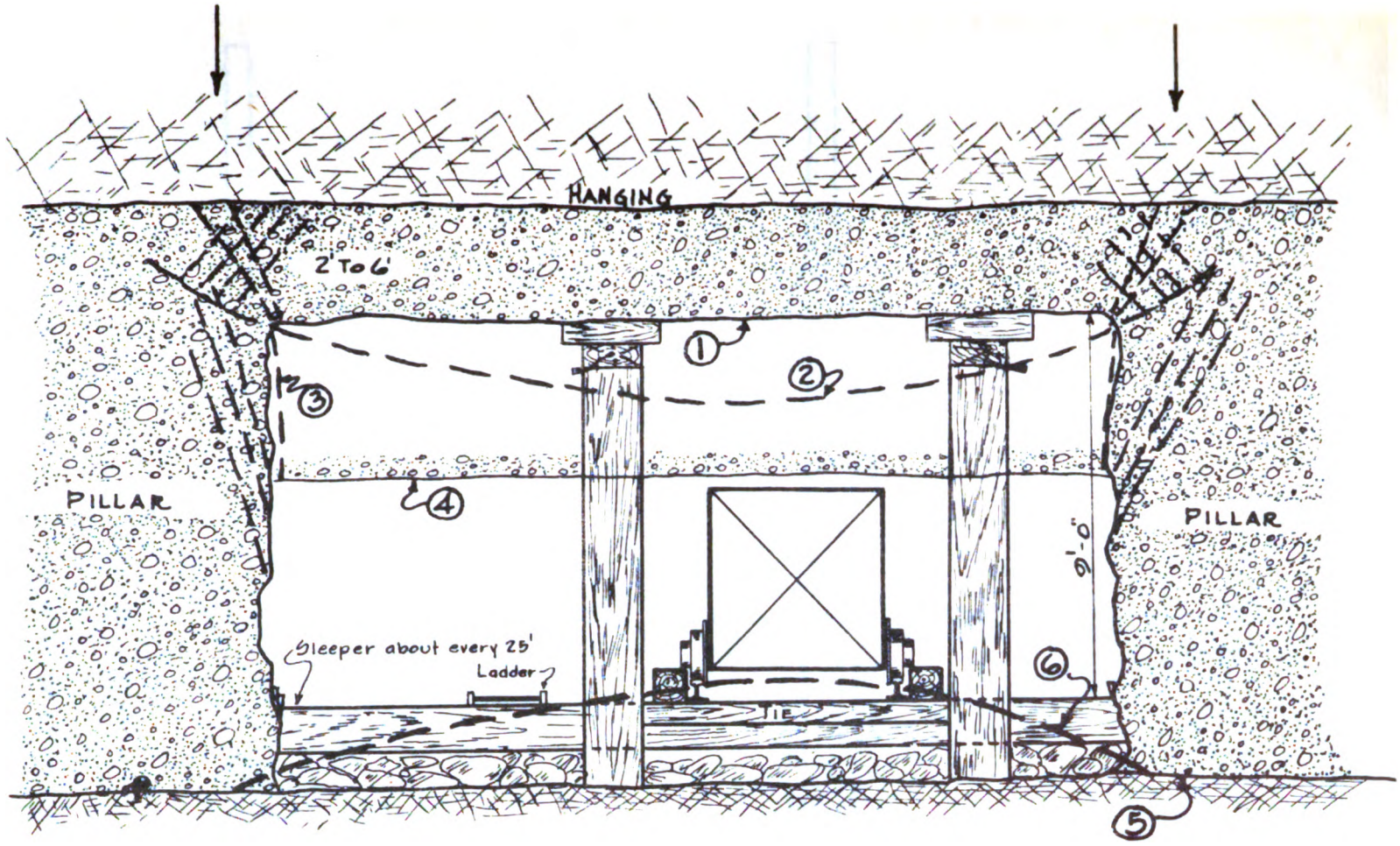
HANGING
2' To 6'
PILLAR
PILLAR
9'-0"
Sleeper about every 25'
Ladder
Tie
① Original Back of Shaft
② Line of Sagged Hanging Wall Before Break
③ Line Along which Hanging Breaks
④ Extreme Position of Sagged Hanging (Here Shaft Repairs Become Necessary)
⑤ Normal Line of Foot
⑥ Bulged Foot Line due to Thrust of Shaft Pillars

Figure 11.— Typical shaft cross section showing lines along which hanging and foot bends take place

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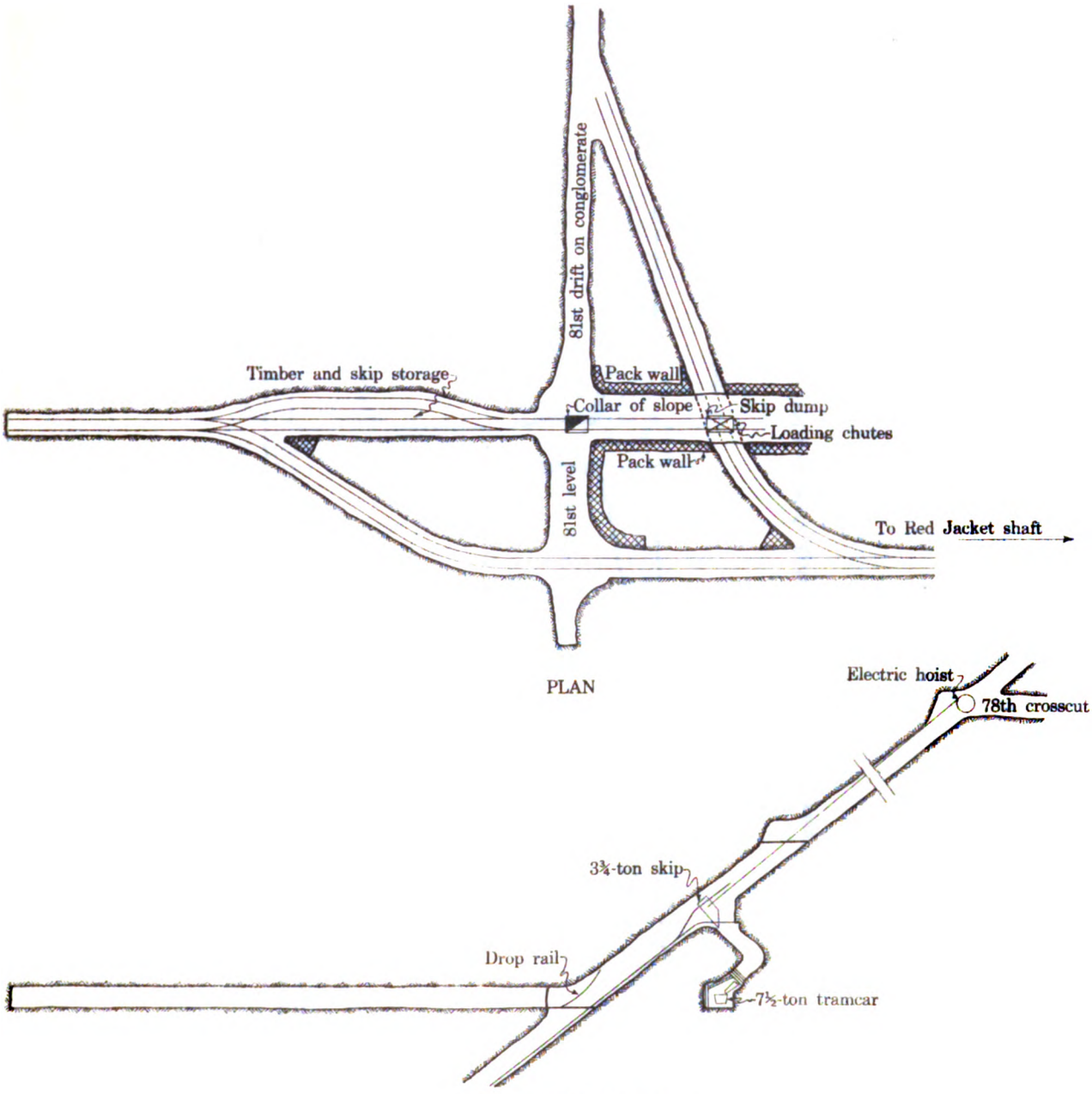

Figure 12.—Collar layout at "C" shaft, showing drop rails

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Figure 13.—Collar layout at "A" shaft, showing electric hoist

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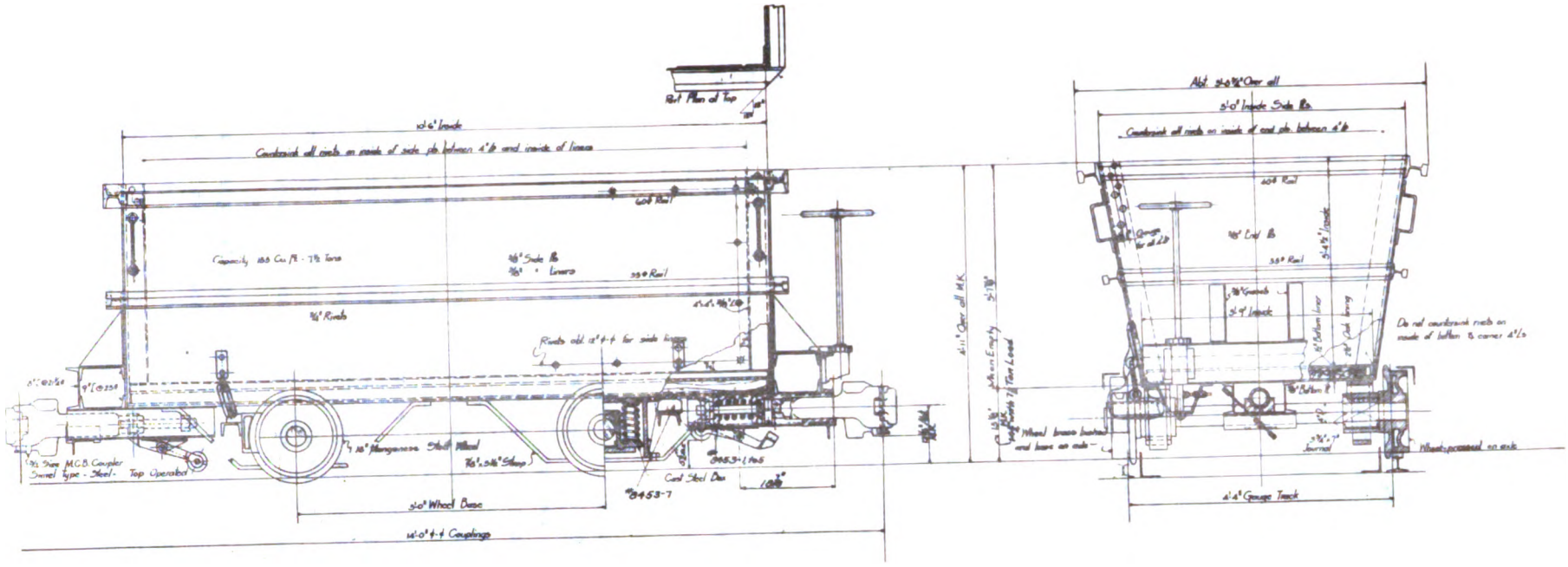

Figure 14.— Tramcar, 7½-ton capacity

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full-capacity production may be maintained at minimum cost up to the time in the future when general "scramming" operations become necessary.

To meet these requirements a through-going tunnel paralleling the conglomerate bed which would give direct access from the Red Jacket shaft to the Hecla and South Hecla workings at the south end was deemed necessary. To fulfill the requirements of safety, permanence, and low upkeep, the selection of a location with reference to the Conglomerate mine workings was of vital importance.

Although failure of the hanging wall rock ordinarily decreases more or less regularly with distance from the conglomerate, movement of hanging in mass results when the formation is cut by a large number of prominent slips along which the movement may be distributed, with the result that the live load is carried to considerable distances above the workings. The footwall was, therefore, considered to offer the only stable ground, and it was decided to drive the tunnel along an amygdaloidal bed which lies 200 feet horizontally below the conglomerate. This bed was found to consist of a soft amygealoid of the fragmental type, barren of copper mineralization, and to be of sufficient width (approximately 30 feet) to encompass the entire cross section of the tunnel (9 by 12 feet) and still leave enough of the bed material overhead so that breaking into the base trap of the overlying flow might be avoided, and the probability of block caving thus averted. The distance below the conglomerate to this bed is sufficiently great to permit absorption by the surrounding traps, of the major pressure resulting from the closing in of stoped-out areas above.

The length of the haulage way, including the enlarged Red Jacket crosscut, is 9,800 feet from the Red Jacket shaft to the south extremity at No. 12 shaft. To facilitate handling timber, drill steel, and other supplies and equipment from this shaft to the haulage way and thence to the incline shafts below, all track gages have been made the same (4 feet 4 inches), and direct connections have been made between shafts and haulage by means of curved "drop rails" (see fig. 12).

The "A" and "C" shafts are equipped with electric hoists located on the 81st level, capable of winding from a depth of 3,000 feet below (see fig. 13). Haulage equipment consists of a motor-generator set with capacity for two 13-ton and one 10-ton General Electric trolley locomotives operating from a 250-volt d.c. trolley. Track is of 50-pound rails laid to a 4 foot 4 inch gage. Tram cars of 7½-ton capacity (fig. 14), solid-body type, and trains of 12 cars are loaded from pockets at the incline shaft heads, and dumped into pockets at the Red Jacket shaft by means of air-operated rotary dump cradles. Locomotives and cars are connected by M. C. B. three-quarter size couplings with tripping device by which single cars may be rotated without breaking train. Seven and one-half ton capacity Kimberley skips operate in balance through the Red Jacket shaft. All trains operate under control of a complete automatic, electric block-signal system, and to control the mine ventilation all doors on the main haulage way are mechanically opened and closed by trains in motion.

Although mining has already extended well below the 81st level and the increasing effect of pressure has had ample time to make itself felt in the immediate vicinity of the haulage, to date no timber or other support has been necessary over a great portion of the haulage opening. The entire length has been given a coating of "gunite" to prevent air leaching of the softer minerals, but in a few places timbering has been necessary.

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DEVELOPMENT BY DRIFTS

Development of the vein from the incline shafts consists in cutting plats (fig. 5) or stations at each 100 feet of incline depth, from which 7 by 8 foot drifts (fig. 6) are driven to the limit of the adjacent mining area - that is, to the property limit or to a pre-established limit between shafts. The timing of drift advance is such that the upper drift of a group shall reach the mining limit before the top stope of a tier of retreating stopes has reached the shaft pillar. This permits the top stopers to move down and start a new stope at the mining limit without loss of time.

RETREATING STOPING

The length of stopes has been consistently shortened as depth was attained. In the original mining operations, stulls were used entirely for timbering stopes; later, due to the increase in vein thickness and quality, square-set timbering was used in all the stoping, and the stoping length was 200 feet. This square-setting was standard practice down to a depth of 5,500 feet along the lode; but as greater depth was attained, and as it developed that the number of stopes running over 20 feet in thickness was rapidly diminishing, and also in order to save the excessive cost of the framed square-sets, a gradual transition was made stull timbering, which is now used entirely for temporary support in all stopes. The stoping length was at the same time cut down to 100 feet. Timbering the lower portion of stopes has become largely a matter of standard practice, but the best judgment of the operating foreman and head timberman is consulted to determine the number and the spacing between stulls set in the upper portion of a stope. Standard practice at the present depths calls for four rows of stulls, in clusters of two timbers alongside each other (fig. 3) and 18 inches to 2 feet in diameter, set with approximately a 7-foot clear space between the near timbers (in the row measured horizontally), and about 7-foot centers between rows measured along the dip of vein (see sketch of stope, fig. 3). These double batteries cover approximately 40 per cent of the height of the entire stope and the practice of placing the double sets never varies up to that minimum height, regardless of how firm the hanging may appear during the early days of the stoping operations. Above this height (about 40 feet) the stulls are placed singly with a variable increase in distance between centers, the actual spacing being governed by the appearance and action of the hanging in each particular case.

Experience has shown that during the time required to complete a stope (approximately 100 days) the maximum thrust of the hanging wall manifests itself at a point about one-third the total height of the completed stope, or just below the fourth double set above the level. This is considered the critical support point in the entire operation, for while it is true that the three double sets below must carry their particular load for a longer period of time, the actual thrust upon them is considerably reduced, due to the proximity of the unmined ground immediately below the level. Set 4 is just below the center of the stope height, 40 feet above the level. It must hold the back for about 60 days after being placed, whereas the next set above (the first single stull to be placed) is well toward the center of stope height and thus above the point of maximum thrust of the vertically imposed hanging load, also the time during which the pressure acts upon it is considerably shorter. Each successive set from this point up, is required for a shorter length of time, and is located under an area of hanging wall which receives great relief of pressure due to the support afforded by the caved-in material of the stope immediately above. Frequently the last 20 to 25 feet require no timber, or at most a few odd stulls placed arbitrarily.

In drilling and breaking the vein at the stope face, three machines and some times four are operated simultaneously in each stope, the number depending upon the speed of extraction

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deemed necessary on account of the pressure conditions in the particular stope in question. The operation may be classed as an "under-slicing" or "bottom-slicing" plan, each slice taking about 6 feet along the entire 100-foot length of the stope. The first slice for a new stope, known as the "cutting-out" slice, is started at the end of the proposed stope nearest the shaft and is extended toward the previously completed stope, which is usually well caved down by the time this first slice reaches it. Each successive slice thereafter is started from the side of the caved stope and extended toward the shaft (fig. 4), thus avoiding the short raises which would be necessary if each slice were started from the shaft side, and also avoiding the dangerous projection which would exist at the caved-stope end of each slice, just before the slice was completed. It is a generally accepted fact here, and it is believed throughout the entire deep-mining world, that such projections present conditions most favorable to rock burst.

It may be well to describe here the manner in which the miner is protected from danger while starting the slice at the edge of the caved stope. Just before each stope is completed to the level above, a double row of stulls known as the "breaker row" is placed from bottom to top, within about 3 feet of the shaft side or within 3 feet of the ground next to be stoped (fig. 3). The row consists of pairs of heavy logs, each at least 2 feet in diameter, set with a space of 3 feet between pairs (measured along the dip), thus leaving a safe area 3 feet wide, into which the hanging may not cave until the new stope has been sliced off. This row is not placed earlier during the progress of stoping, for the reasons that approximately 3 months is required to complete each stope and that the average life of the timber used, principally hemlock, maple, and birch, is from 4 to 6 months. Therefore from the time of completing one stope to that of completing the next, the major portion of the useful life of the "breaker row" has been expended, which is the end to be desired. As the last slice at the top of each stope is cut, the floor arch of the level above is drilled, preparatory to blasting (figs. 3 and 7), but the actual breaking is deferred until the slice has advanced sufficiently to leave a short length of the protecting arch above the driller. The stope above is by this time completely crushed in, but the danger of a sudden rush of caved material into the stope below has been obviated by a wall of heavy lagging timber, laid horizontally along the upper side of the "cutting-out" stull row at the time of completing the upper stope (fig. 3). This lagging is placed as the last timber operation in all stopes, anticipating the recovery of the floor arch from below.

The successive recovery of each stope on a particular level is a retreating operation from a preestablished boundary between adjacent shafts, toward the extraction shaft for the area in question. A complete mining operation on either side of a producing shaft is a series of four retreating stopes on four successive levels, the top stope leading the next below by approximately 150 feet, and so on down (fig. 4). This lead is considered a minimum distance over which the floor arch should remain intact during the active life of the stope above, and it also permits a lapse of time more than sufficient to consume the useful life of the timber supports in a particular stope before the floor arch removal operation reaches it, thus relieving the pressure on the arch at the time of removal.

No attempt is made to speed hanging collapse in stopes by blasting out or otherwise attempting to remove timber supports. Any regulation of the time of collapse must be made by increasing or decreasing the number of stulls placed during the active stoping operation.

As mentioned previously, experience of the past has demonstrated that points at which so-called "air blasts" or "rock bursts" are more likely to occur are at projections or points of rock, such as intersections of drifts and crosscuts, turnouts, entries to engine

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rooms, projecting points on stope faces, etc. It is unavoidable that such projections must be made in laying out a complete haulage system, together with space for motor generators, transformers, fans, pump rooms, etc. In deep mines where pressure of the overhanging load reaches rock-burst proportions, it is also unavoidable that such points of projection must be maintained to keep at a minimum the span of unsupported back in such places as must be kept open for an extended length of time. Timber sets of long-life material such as fir or pine are of prohibitive cost and require frequent replacement. The support which has given the greatest satisfaction on the Calumet conglomerate lode, in life, cost, and safety, consists of a "pack wall" or built-up block of "dry wall" with interbedded layers of flat timber, the timber laid horizontally and built up with the broken rock; about one-sixth of the total cubic volume of a block consists of timber. The first cost of construction of this "pack wall" for a given area of hanging is the same as that of supporting a similar area by means of stulls of short-lived timber, such as in stopes. The labor used in placing the rock-timber block is double that for placing stull timber, but the cost of material offsets the high labor charge. Where this form of support is used, the point of rock is first removed entirely and the rock-timber "crib" is built up in its place. As the "squeeze" takes place over such cribs, the rock is forced tightly against and into the timber layers, cutting off circulation of mine air and materially preventing timber decay, with the result that the useful life of the block is several times that of any timber support which might be used. In maintaining the sides of inclined subshaft heads where severe slabbing has taken place, such cribs have been built up in the form of walls from 6 to 10 feet thick, along the dip of the shafts (about 36°) and have proved of inestimable value, due to their ability to cushion the thrust and compress very slowly, giving ample warning of crushing pressure on the timber of the shaft-way with the result that time is allowed to replace crushed headers on the shaft sets before the sets themselves have been ruptured.

Under the longwall retreating system adopted for the deep Conglomerate workings, bursts have become quite a rare occurrence and are not looked for except in the last or next to the last stope at the shaft pillar. In removing the shaft pillar, although the adjacent stopes have by that time caved tightly, it may be said that almost every form of pressure result is encountered, such as wedging or the breaking of large masses from the pillars, heaving of the foot, back subsidence, and excessive drag. Pillar removal practice is indicated by Figure 8.

METHODS OF SAMPLING

No sampling of ore in place or in muck piles is practiced in the copper country of Michigan because of the difficulty of sampling ores containing copper in the metallic form in such irregular distribution and in pieces of considerable size. It is not believed that results approaching reasonable accuracy could be attained by sampling either the amygdaloid or conglomerate lodes of this company. In formations such as the porphyries where the copper occurs in minerals of brittle or earthy character, sampling in place produces results of considerable accuracy.

The general concensus of opinion among mining men in the Lake Superior district is that mine sampling does not pay. Mill-feed sampling is not practiced generally, but in special cases may be used to make a rough estimate of grade of ore from a certain section.

In past years this question has continually turned up for discussion and decision, and on a great many occasions the attempt has been made to prove the practicability of sampling, with questionable results at all times, and in some particular cases with results so misleading as to constitute a menace rather than a help. Consistent diamond drilling into the footwall at regular short intervals, in the case of a mine where copper was known to

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occur in very profitable patches in the footwall, proved very expensive to one company because of the misleading character of the information.

The best method of determining what is and what is not ore is to keep the section foreman in constant touch with the assay value of the mill product coming from his branch, by written reports to him. He then possesses a daily knowledge of what all stoping faces show in the way of exposed native copper and its continuity and degree of concentration; this data, combined with his report of mill results, gives him a criterion for judging the grade of ground of a certain appearance.

On his daily trips into development openings, he notes the condition of the face and estimates its grade. At the end of the month, he combines his notes of daily estimates and arrives at a composite estimate of grade developed by the month's work. Maps are made up to show the estimates by months and are checked against mill returns when this area contributes ore to the mill. These checks are, of course, quite general and must be made by engineers familiar with field conditions.

Owing to the great linear extent and depth of the operating mine and the care with which past records have been kept, it is possible to gage (without sampling) the grade of ore now exposed and also that to be developed in the immediate future with a degree of accuracy not to be attained by sample assays.

Estimation of Tonnage and Value in Mines of Calumet and Hecla

Estimates of tonnage extending beyond the limits of complete development and the grade assigned to such extensions are carefully and conservatively computed and have been subjected to repeated check by actual mill extraction records over a period long enough to have brought the procedure to a state of high reliability. Each step is based on conservative estimates before it is introduced into the combined computation. All geological indications are carefully considered, and only such extensions of the ore lines are made as can be safely counted upon for the future on the basis of the obvious tendencies of the vein at the time.

The thickness and perimeter of the stoped areas are carefully recorded and mapped. The exact area of vein mined over is scaled from the maps, and with a knowledge of the mill returns for various areas, which are separately concentrated, computations are made showing the copper content of the ore coming from a particular area and also the number of pounds of copper per unit of area cleaned up. In the development openings, which are continually being advanced at greater depths on the lode than the average depth of stoping operations, frequent crosscuts are made to hanging and footwalls, to determine in advance of stoping operations any tendency of the workable vein to widen of pinch. Mineralization for a particular vein may show a tendency to improve or decrease as greater depth is attained, due either to a change in the form of distribution throughout a given area or to a change in the thickness of vein throughout which the same form of mineralization may be distributed. Such changes are usually not precipitate, however, and lacking the sudden appearance of a major fault, cutting off vein mineralization, the tonnage and grade which may reasonably be expected can be estimated with considerable accuracy for a number of years in advance of actual operations The probability of the sudden appearance of major faults, such as mentioned, can be anticipated with a fair degree of assurance by a careful study of any unusual undulations appearing on the survey maps of advanced development openings. The long record of the district as a whole indicates that such occurrences are very unusual and are less likely to appear as depth is attained.

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Ventilation

Due to the high rock temperature in the lower portion of the mine and the relatively low average surface temperature, all primary ventilation is accomplished entirely through the medium of natural air currents.

From the records of the United States Weather Bureau, covering a period of 20 years the average mean surface air temperature is 40.5° F., dry bulb, the average mean relative humidity for the same period being 82 per cent, corresponding to an average mean wet-bulb temperature of 38.2° F. At a vertical depth of approximately 5,502 feet virgin-rock temperature measurements recently taken show a temperature of 93.9° F. on the 93rd level. This indicates a rock temperature gradient of 1° rise for each 103 feet of vertical depth from surface. The gradient from 4,800 to 5,500 feet vertically (81st to 93rd levels) by the most recent tests appears to be 1° for each 100 feet. All temperatures were obtained by placing three standardized mercury thermometers, calibrated to tenths of a degree, in holes 10 feet deep drilled for the purpose in the side walls of advancing development openings as closely as possible to the advancing face. The collars of the holes were blocked to air currents while thermometers were in place, and periodic readings were made on all three over a sufficient period of time to avoid the possibility of including any effect of the heat of blasting in the finally accepted virgin-rock temperature. In some cases in order to gage the possible effect of air cooling due to the exhaust of drilling machines, the holes were drilled at the end of a short crosscut, which was bratticed off from the advancing development opening.

The great difference in rock temperature between surface and mine workings produces a difference in air densities in different parts of the mine, causing a natural air flow, the major currents of which remain fairly constant in direction throughout the year, because of the comparatively small effect that changes in surface air temperature have upon the average temperature of air columns of such great length. No great difference of collar elevation exists for any of the shafts throughout the mine. It is particularly noticeable in natural-draft mines that the larger volumes of fresh air tend to go directly to the lowest openings, due to the rarifying power of the high-temperature zone. In the ventilation of the upper workings this is largely compensated for by large openings between shafts, such as drift stopes and stoped areas, presenting low resistance to the flow of air. As all stopes retreat from boundaries toward the slope shafts, extracting floor pillars as they go, a large volume of air circulates at very low velocity through the caved material of the stopes. The airways of the mine, as compared with the average deep mine, are large and kept free from obstructions. All shaft timber is placed in good alignment and drifts and crosscuts are usually free of timber. Where openings are kept up by "pack walls," the wall face is well aligned and smooth. The following table gives the analysis of mine-air samples taken by G. E. McElroy of the United States Bureau of Mines and shows the air to be of good quality throughout the mine.

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Air-sample analyses - Conglomerate mine of the Calumet & Hecla Consolidated Copper Co

Bottle No.	Laboratory No.	Date, 1928	Time, a.m.	Location and description	Volume of air, c.f.m.	Chemical Analyses			
						CO_2	O_2	N_2	CO
381	48,537	7/11	9:10	Return air from Red Jacket slope section on 57th level	7,000	0.15	20.80	79.05	0.00
383	48,536	7/11	9:50	Face of 58 S. stope off Red Jacket slope	2,500	.11	20.76	79.13	.00
380	48,534	7/11	11:15	Face of 64 N. (abandoned) drift off Red Jacket slope	None	.22	20.36	79.42	.00
379	48,535	7/12	11:35	Face of 92 S. drift 100 feet off "C" slope	Drilling exhaust	.10	20.85	79.05	.00
391	48,538	7/12	11:50	Face of 91 S. drift 700 feet off "C" slope	Drilling exhaust	.12	20.83	79.05	.00
390	48,539	7/13	12:30 p.m.	Return air from stopes, on 80 drift 2,000 feet north of No. 6 Hecla shaft	7,000	.09	20.78	79.13	.00
385	48,591	7/14	10:00	No. 5 Tamarack upcast shaft at 29th (71 R.J.) level	37,000	.05	20.92	79.03	.00

Every effort is made to keep all downcast shafts as dry as possible in order to reduce the relative humidity of the air at the working places.

At a vertical depth of 4,750 feet, at one of the downcast points, dry-bulb temperature is estimated to be about 14° below virgin-rock temperature for this depth. Measurements at this depth at No. 12 downcast, show 64° wet-bulb and 72.5° dry-bulb temperature, corresponding to 63 per cent relative humidity. For such a depth this is considered remarkably cool, dry air and is considered to be due in part to the dry condition of the shaft, but to a still greater degree due to the fact that the mine water contains a high content of natural calcium chloride, which materially reduces its rate of evaporation.

Auxiliary ventilation of development headings and dead ends is taken care of by electrically driven compressed-air blowers. Fan-pipe installations driving large volumes through short distances are used in ventilating deep pump and hoist rooms.

In the two main pump rooms at No 5 Tamarack shaft, where a large volume of heat is generated by three 550-hp. motors in each room, single-inlet fans are used having 3-foot diameter rotors 1.5 feet wide, operated at 750 r.p.m. These each deliver 30,000 cubic feet of air per minute from the rear of the pump rooms to the shaft.

DRILLING

Machine drills of the Leyner type with 3½-inch pistons, mounted on 3-inch single jack posts are used entirely for stoping and drifting and are operated under 80-pound air pressure. Jackhammers are used only on special work, such as trimming in places and blockholing generally. Hollow hexagonal drill steel, 1 inch across flats, and with 1/4-inch holes is used throughout, a set consisting of a 2-foot starter and drills varying by 1-foot lengths from 3 to 10 feet, inclusive. Gage of bits is reduced 1/16-inch for each increase of 1 foot

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in drill length; the starter bit is of 1 3/4 inch size and the 10-foot finisher is of 1 5/16-inch gage. Crossbits are used entirely because of the tendency of single bits to jamb on account of the cracks encountered at the face in the deep Conglomerate mine workings. The placing of holes in stoping slices is left largely to the miner, supervised by the shift boss, and varies considerably because of the changeable nature of the face due to pressure slabbing or to jointing. In drifting, the pyramidal drawcut is standard practice, a cut requiring usually 19 to 20 holes, 6 feet deep, to give an advance of 4 1/2 feet per round (fig. 6).

BLASTING

All blasting is done at the end of the working shift. Powder is delivered to each miner once a week and stored in metal boxes provided with locks. Caps and fuse are kept in separate metal containers at some distance from the powder. The powder sticks are 1-1/8 inches in diameter, 8 inches long, and consist of 30 per cent nitroglycerin for old backs, 40 per cent nitroglycerin for shaft pillars and stopes, and 70 per cent giant gelatin for drifts. In charging, the primer is usually set near the bottom of the charge and firing is done by fuse and detonators. Timing of firing is determined by the length of fuse cut for the particular hole. For special work, such as long raises and vertical shaft sinking, electric delay blasting is used with great success.

TRAMMING

The tendency of the conglomerate rock when blasted is to break into slab-shaped pieces, so that in spite of the relatively flat vein dip, the greater portion of the material broken in stopes runs to the bottom. In shaft-pillar removal, timber chutes are built at the stope bottoms for car loading, but ordinarily for the retreating stopes rock is scraped from the drift floor up a scraper incline of sufficient height to permit the tramcars to be run underneath and filled by ore dropping through an opening in the scraper platform above (fig. 15). About 15 per cent of the stope production coming from the upper stope faces is scraped down to the level by the same scraper that is used to fill the tramcars on the level (fig. 16).

To accomplish this, a "snubbing-post" and head pulley are set at the top of the stope, and the scraper drag line after passing through a pulley on the hanging-wall side of the level is diverted to the head pulley. After cleaning out the stope, it is a simple matter to replace the head pulley on an inside stull and resume scraping along the level.

All mine cars employed in development work and stoping are of 3-3/4-ton capacity and of the end-dump type, having 4-foot gage. The haulage, except on the 81st level, is by storage-battery locomotives, two types being used: 3-ton, single-motor, 48-cell MV13 Exide batteries; and 7-ton, single-motor, 48-cell MV27 Exide batteries. Each of these are single-wheel drive through double-reduction gears.

The motor-generator sets are of two types: 40-kilowatt, 125-volt, d.c., driven by 60-hp. motor; 20-kilowatt, 125-volt, d.c., driven by 31-hp. motor.

PUMPING

Glacial drift running from 10 to 40 feet in depth covers the Calumet area, and while it is not possible to drain this deposit thoroughly over the entire area above the mine

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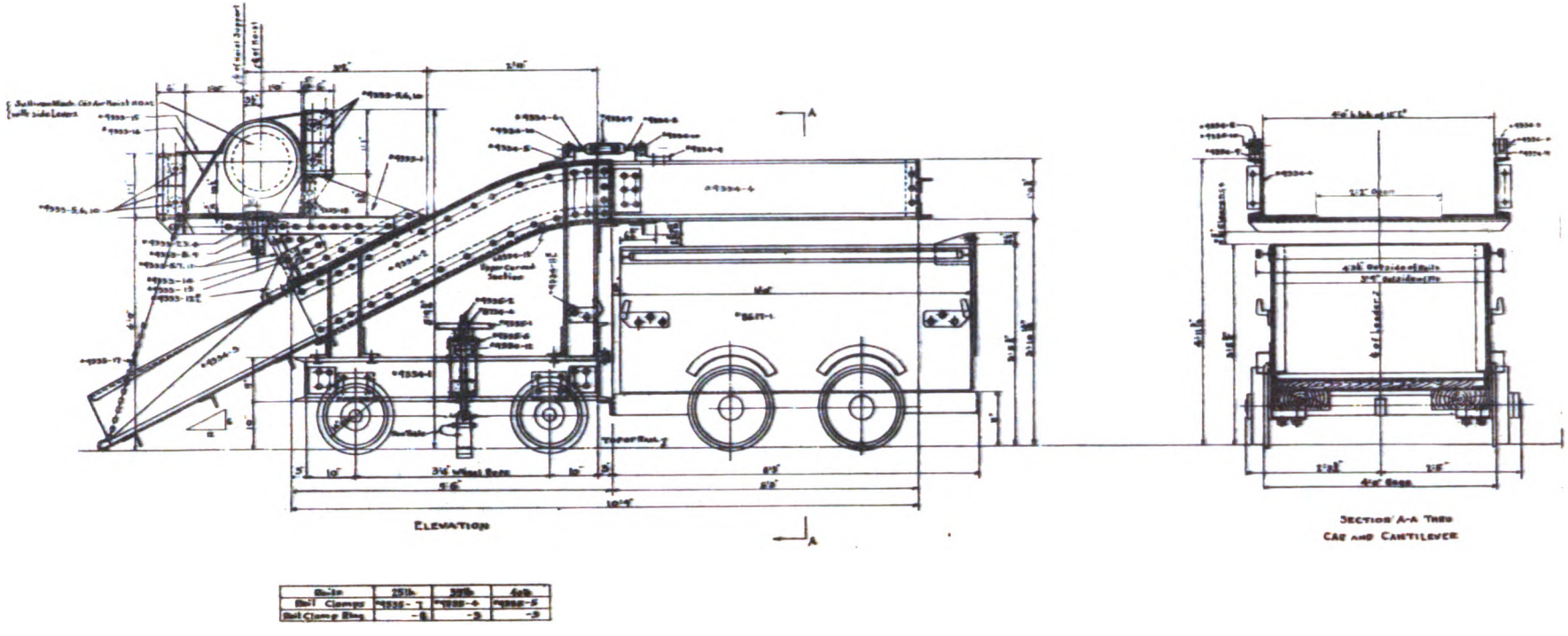

Figure 15.— Arrangement of scraper slide

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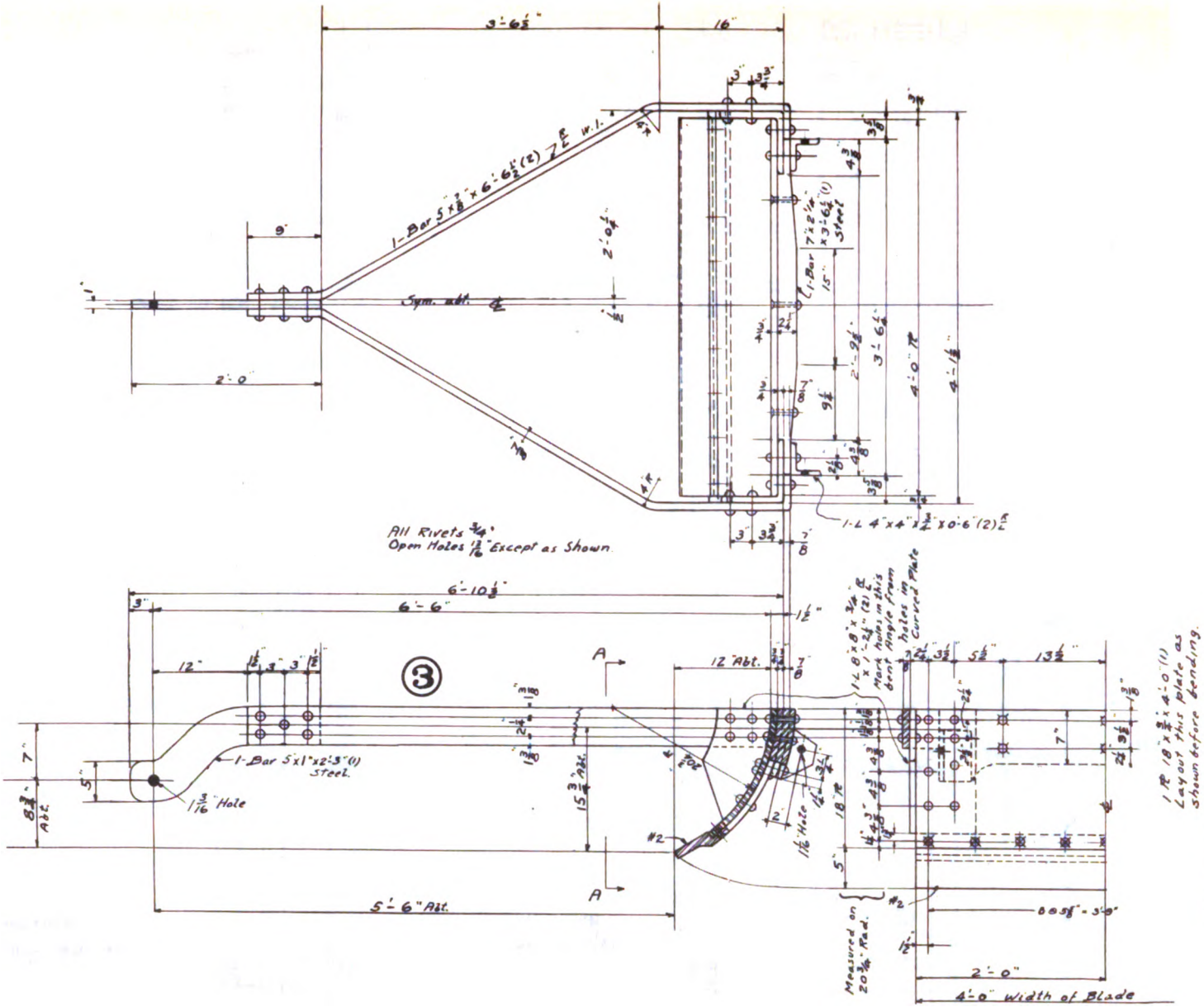

Figure 16.— Stope and drift scraper

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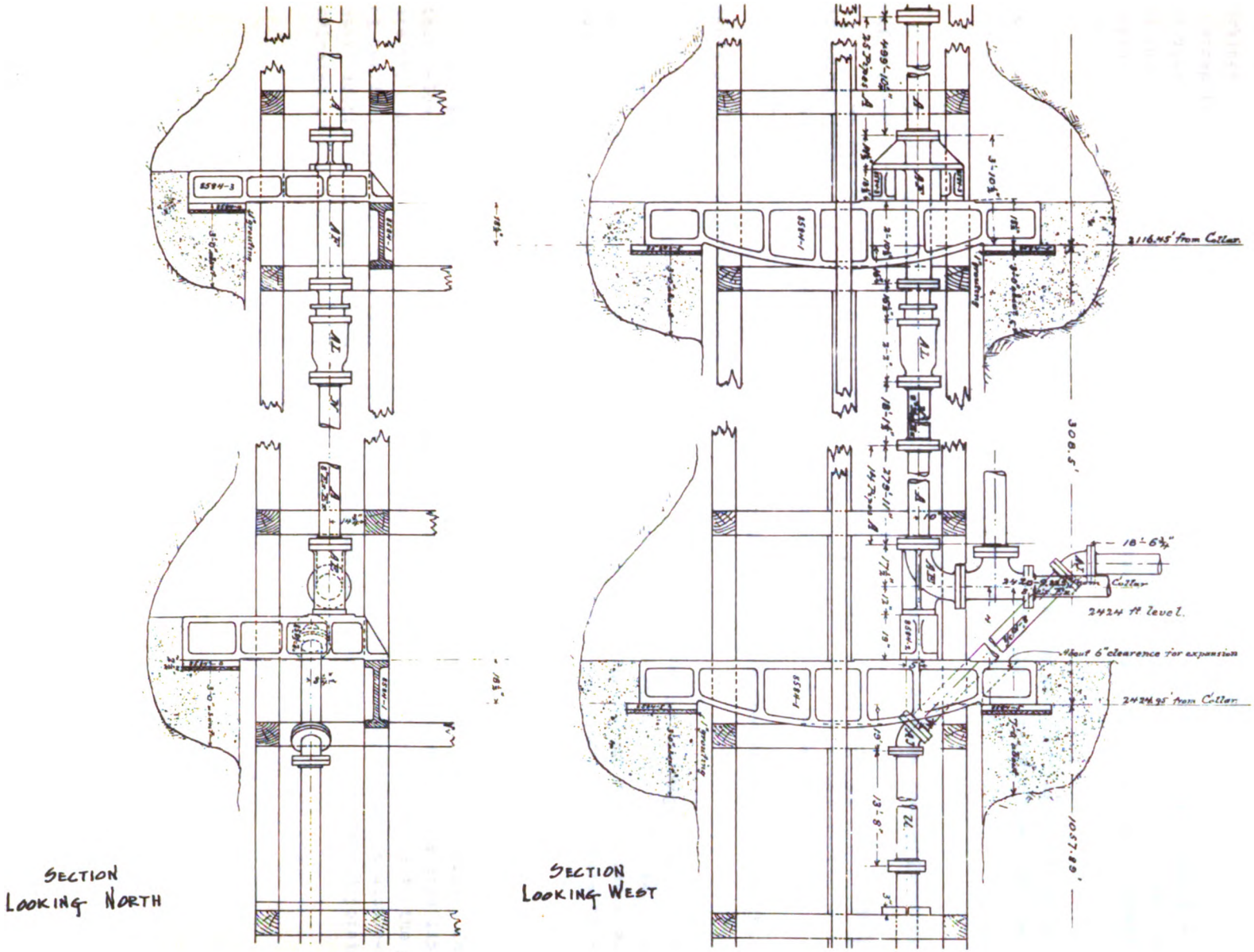

Figure 17.– Column pipe and supports at No. 5 Tamarack

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workings, a considerable saving in mine pumping has been accomplished by the use of surface intercepting ditches and surface pumps. So-called "collar" pumps are placed at points nearest under the shaft collars where sufficient water can be caught to justify a separate pumping unit. Somewhat more than 2,250,000 gallons per day are pumped to surface by the combined pumping facilities in the Calumet conglomerate lode workings, the principal installation being located at No. 5 Tamarack vertical shaft.

This plant consists of two pump stations, one 2,424 feet vertically below surface, the other 5,156 feet down or 2,732 feet below the upper station to which it delivers water in one lift. At each station the equipment consists of the following:

3, 334-g.p.m. vertical triplex plunger pumps.

3, 350-hp motors, using 440-volt a.c., starting on by-pass accomplished by one-half voltage or 220 volts taken from transformers.

1, Air compressor, triple-expansion, delivering air to air chambers at 1,250 pounds per square inch.

3, Oil transformers, 500-k.v.a., step down from 4,000 to 440 volts.

1, Switchboard.

1, Fan, single-inlet, rotors 3-foot diameter by 1.5 feet wide, 30,000 cubic-foot capacity, driven by 75-hp. motor.

Water is supplied to the lower pumps from a large-volume storage sump and relayed to a sump of nominal capacity just off the 2424-level pump room. The shaft water column consists of 8-inch diameter steel pipe, heavily galvanized inside and out. Flange connections are used having male and female joints. During construction all pipes were threaded, flanges were screwed on, flange bases were welded to the pipe and then the welding strip was galvanized. Thus no threads or bare steel were left exposed to the corrosive action of the mine water. All pipes except short "filling-in" lengths are 20 feet long and the columns are supported in the shaft by heavy cast-steel girders, at intervals of about 300 feet (fig. 17). The girders rest on concrete piers, carried in hitches cut into shaft walls. Immediately below each girder, expansion joints (fig. 18) have been provided for the pipe line, to care for changes in pipe length which might occur during periods of interruption to the pumping service. The pipe and flanges used are of various weights from standard to double-extra heavy, according to the position occupied in the column. Details of supports and joints are indicated in the figures.

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Table 1.- Summary of costs in units of labor, power, and supplies

Name or number of mine: Conglomerate — Period covered: Year 1930.

Ore tonnage: Mined and hoisted, 872,834; development, 40,964; mining, 831,870.

Mining method: Open stopes with temporary stull supports; retreating system.

	Development	Mining (stoping)	Total
A. Labor (man-hours per ton):			
Breaking (drilling and blasting)	1.413	0.386	0.434
Timbering	.308	.481	.473
Shoveling	1.077	.397	.423
Haulage and hoisting	-	-	.248
Supervision	-	-	.138
General	-	-	.890
Total labor underground	2.798	1.264	2.612
Av. tons per man per shift	-	6.328	3.062
Labor, percentage of total cost	-	-	69.84
Av. tons per man-shift on surface properly chargeable to underground operation	-	-	7.23
B. Power and supplies:			
Explosives:			
Pounds per ton	-	-	.75
Kind and grade	-	-	30-40 and 70%
Timber, linear ft.	-	-	6.192
Power, kw.h. per ton:			
(1) Air compression	-	-	4.942
Other supplies in percentage of total supplies and power	-	-	38.32
Supplies and power, percentage of total cost	-	-	30.16

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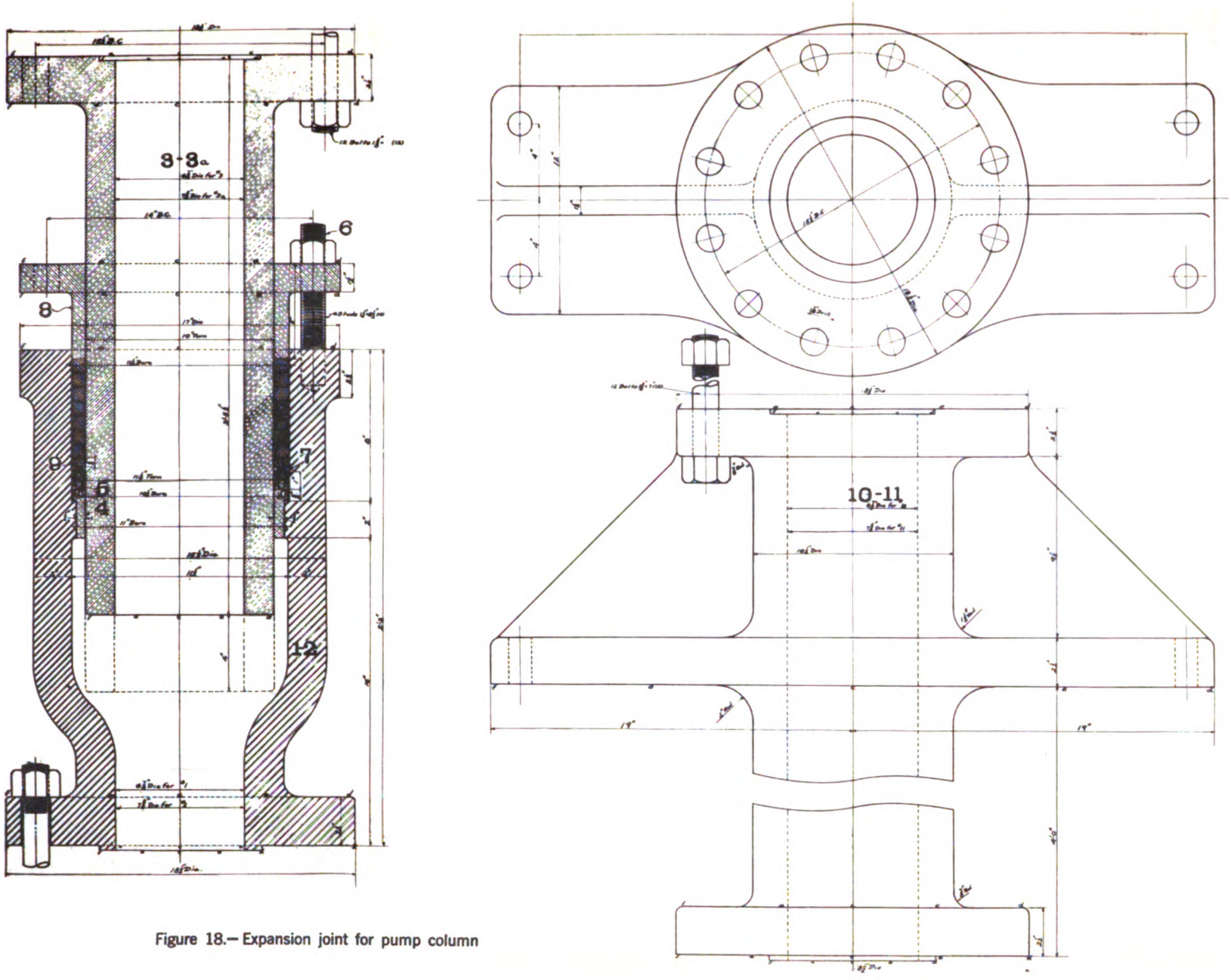

Figure 18.— Expansion joint for pump column

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